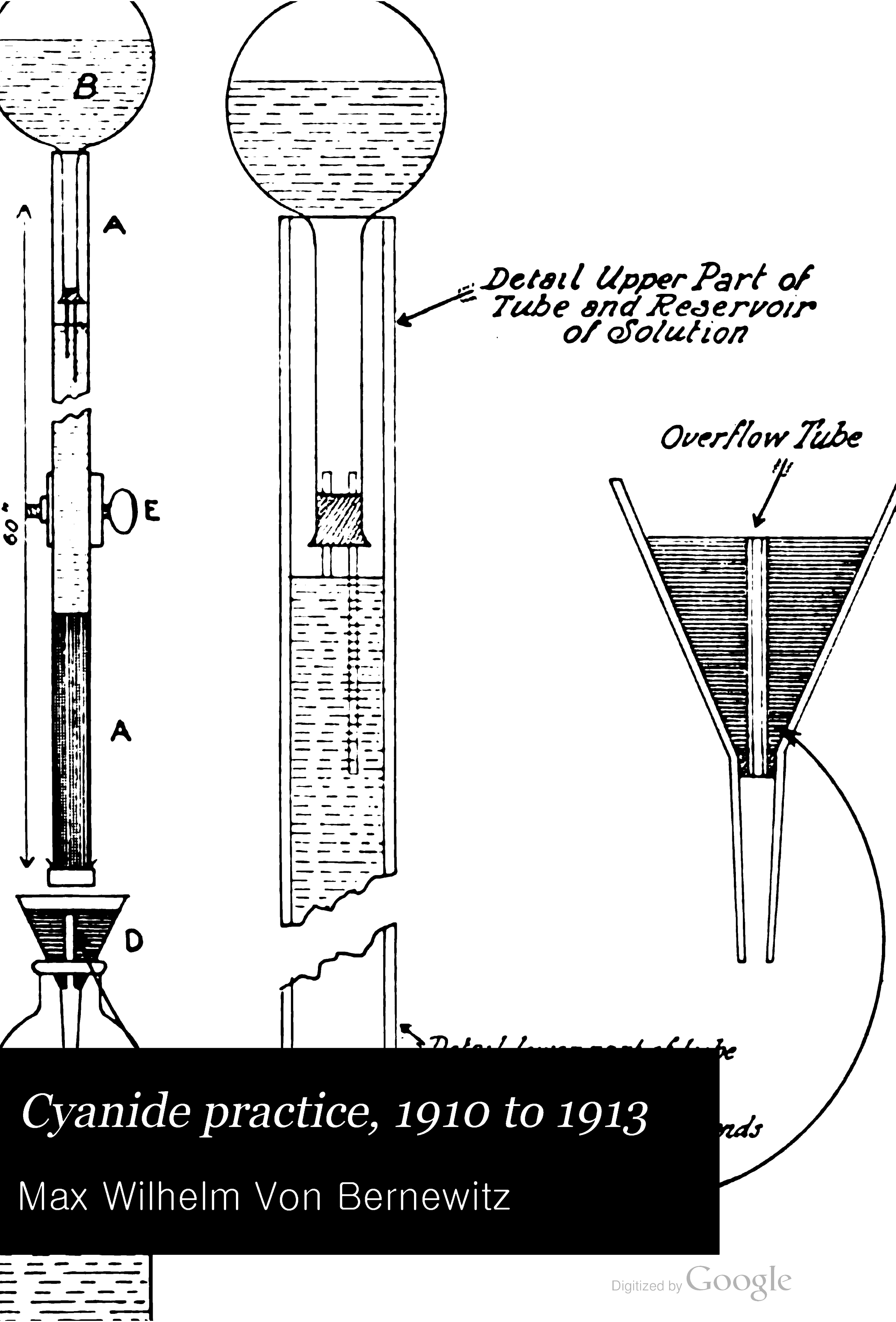
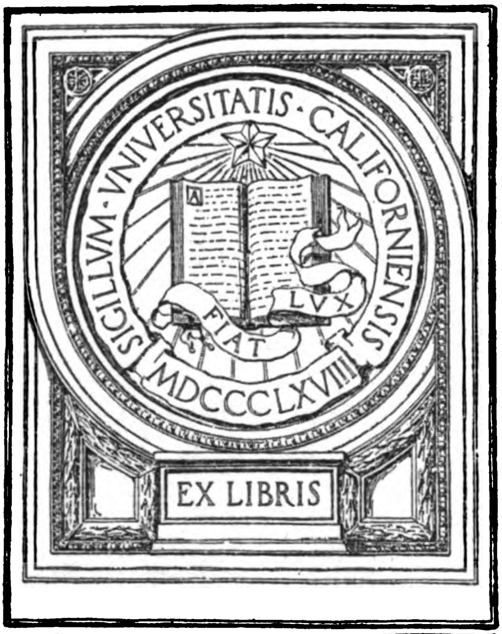


Cyanide practice, 1910 to 1913

Max Wilhelm Von Bernewitz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CYANIDE PRACTICE

1910 TO 1913

EDITED BY
M. W. von BERNEWITZ

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AND
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PREFACE

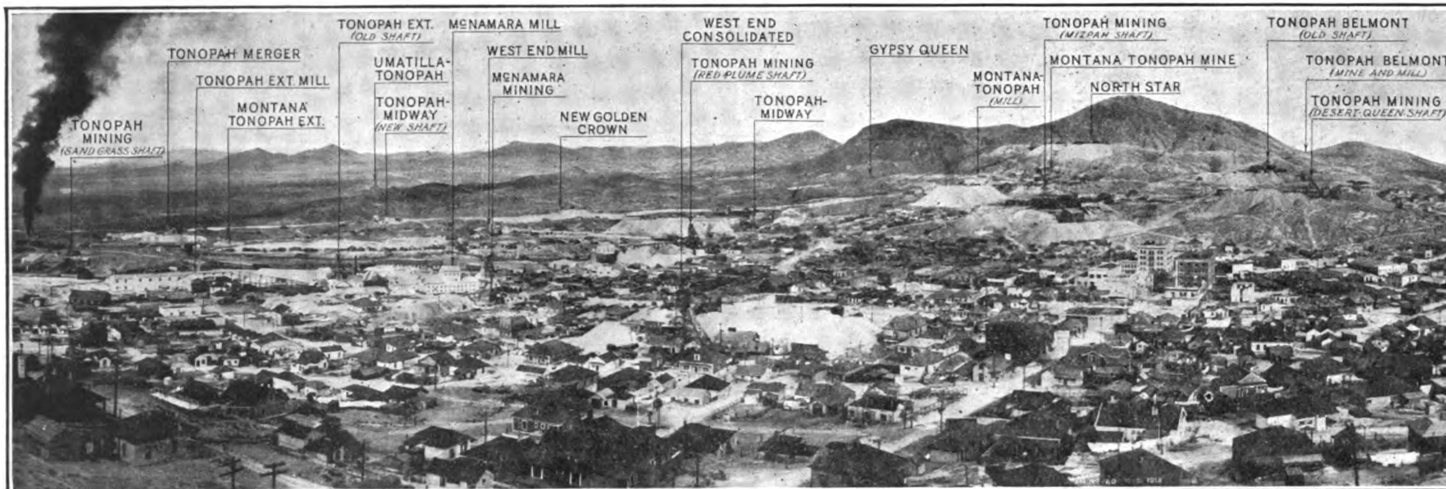
This is the third in the series of books that began with 'Recent Cyanide Practice' edited by Mr. T. A. Rickard, and printed in 1907, and which was continued by 'More Recent Cyanide Practice' edited by Mr. H. Foster Bain and printed in 1910. The plan of all three books is the same. They include articles on all phases of current cyanide practice based upon experience in all parts of the world. In the main the papers here reprinted were first published in the *Mining and Scientific Press*. One or two related intimately to discussions in the columns of the *Press*, have been taken from *The Mining Magazine*, and the *South African Mining Journal* to which acknowledgment is gladly made here and in the body of the text. In preparing the various papers for use in this book a number have been materially condensed and my own have been revised. Instead of the articles being arranged according to dates, they are given under their proper heads, from the chemistry of the process to the clean-up, although the dates on which they were published are also given. This arrangement should find favor with readers, who will be able more readily to find any particular department or process. The time covered by the articles here brought together extended from July, 1910, to January, 1913, except that a few appearing in the *Press* in the early part of the present year have been included. The necessity of restricting the size of the volume has made it necessary to omit a number of excellent papers of local or temporary importance, and also many that deal especially with ore dressing and stamp milling rather than cyanidation proper. A number dealing especially with cyanidation in Mexico having already been printed in Mr. Ferdinand McCann's work 'Cyanide Practice in Mexico' have also been left out. Even with these omissions, this book will, I hope, serve as have its predecessors, the main purpose of furnishing in convenient form a general view of the best in current cyanide practice.

M. W. von Bernewitz.

San Francisco, September 1, 1913.

LIST OF ILLUSTRATIONS

TABLE OF CONTENTS



TONOPAH, NEVADA

From the standpoint of geology, mining, and metallurgy, this busy centre, situated at an altitude of 6500 ft. on the desert of Nevada, is full of interest. The ores contain silver and a little gold, and bullion is shipped away by the ton, the average monthly yield being about 35 tons. The total output to June 30, 1913, was worth approximately $63,200,000, and $18,012,722 was paid in dividends. Eleven producing mines and several in the development stage contribute to considerable activity. Oil and electric power from California have helped to lower costs. Simplicity of operation is the keynote in the modern stamp-mills and cyanide plants.

HISTORY OF THE CYANIDE PROCESS

By J. McCombie

(June 24, 1911)

Although the cyanide process was born in Glasgow, it was cradled, reared, and brought to an advanced stage of perfection at Karangahake, New Zealand, where it was in full operation for nearly a year before its adoption anywhere else. To the Crown Mines Co. belongs the credit for introducing this process, in a practical way, to the mining world, early in the year 1889, almost immediately after its development by J. S. McArthur and his colleagues, R. W. and W. Forrest. It was brought about in this way: Thomas Melville, an old Auckland resident, floated the Crown Mines Co. in Glasgow, and some of the ore which he took home for flotation purposes was treated by the embryo cyanide process with highly satisfactory results. Arrangements were then entered into with the McArthur-Forrest company to build a plant and treat Crown ore at the mine. Pursuant to certain conditions a staff of hands was sent direct to the mine from the experimental plant, Glasgow. The members were: John McConnell, manager; Frederick Smeaton, assayer; James Tegart, cyanider; William Dempster, engineer, and Peter McFarlane, carpenter.

I give the names of these men here because there is a strong disposition on the goldfields of the Auckland province to belittle the pioneers in every branch of the gold-mining industry, and to relegate such men to obscurity. Owing chiefly to want of good accommodation for the transport of machinery, there was a long delay attendant upon the completion of the treatment plant at the Crown mine, and a small plant was erected in the Woodstock furnace-house, in consequence. This plant comprised one breaker, one Lamberton mill—dry-crushing—two vats, fitted with mechanical agitators, one stock solution vat, one sump vat, one filter-press, and four small barrel towers, for precipitation purposes.

Briefly, the process was fine grinding, agitation, filter-pressing, and precipitation on zinc shavings. The first parcel of ore dealt with was taken from the Maria vein in the Kenilworth mine, of which I was then part owner, and manager, and which is now included in the Talisman company's property. The plant was started early in the month of June, 1889, and I have now before me a copy of the first treatment sheet:

VALUE OF ORE BEFORE TREATMENT.

Gold	£4 0s. 0d.	per ton.
Silver	£1 19s. 0d.	" "
Total value	£5 19s. 0d.	" "
Recovery, gold	89.2	per cent.
Recovery, silver	70.9	" "

The strength of solution used was 0.5%, and the consumption of cyanide was 4 lb. per ton of ore treated.

After a series of experiments with both wet and dry-crushing, dry-crushing in Lamberton mills was decided upon, and the ore was treated by upward percolation, with subsequent water washes, pulled through each charge by means of vacuum pumps. This simple process more than realized expectations in the case of the Crown ore, and it was generally adopted throughout the Dominion, where it remained in use in a good many mills for several years. As depth was attained in the mines the ore became more refractory, resulting in a high percentage of soluble sulphates being formed in the drying furnaces, or kilns, and the consumption of cyanide became a serious item in the cost treatment. Then mill after mill went in for wet-crushing, with dilute solution of cyanide in the mortar-boxes, and the first man to give the new departure a practical test, with 20 stamps, was H. H. Adams, at Waiorongamai. Except in the case of the Crown Mines the system was not continued, and the process gradually veered round to ordinary wet-crushing, amalgamation, concentration, and hydraulic classification, in the order named, followed by percolation for the sand, with agitation and decantation for the slime.

This brings the history of ore treatment by the cyanide process up to date, when decantation is being rapidly superseded by filter-pressing, and a good many New Zealand mills now hold a front rank position in the matter of ore treatment. It seems strange, however, after trying so many variations of the cyanide process, that all have found it necessary to return to the starting point, by adopting the system first introduced at Karangahake, in the year 1889, by the patentees. Today the tendency is in the direction of abolishing amalgamation and concentration, and in the mill of the future there will be only one system of treatment, which will be confined to the cyanide process throughout, regardless of the character of the ore, provided it is all ground to an impalpable paste and dealt with subsequently by agitation and filter-pressing.—*Auckland Herald.*

CHEMISTRY OF CYANIDATION

THE CLANCY PROCESS

By J. C. Clancy

(December 31, 1910)

*The Clancy process is designed especially for the treatment of refractory ores, in which the gold is associated with chemical compounds and is not susceptible to the action of solvents until the ore has been roasted, or until the gold has been disassociated from the chemical combination by some other method of oxidation. A long search for a suitable solvent for such ores, led, after repeated disappointment, to discovery that urea, in conjunction with cyanate, and electrolysis of the solution, dissolved gold leaf. The next question was, would any similarly constituted organic compound do the same thing. Urea being an amide compound (carbamide), I thought at once of cyanamide. I thereupon added a solution of cyanamide (using the calcium cyanamide of commerce which is soluble to the extent of about 58 to 65% in water) to a cyanate solution, electrolized it, tested the solution as before with gold leaf, and dissolution of the gold took place in a few minutes. With a view to studying this peculiar reaction, I tried electrolysis on a solution of calcium cyanamide, without any admixture other than the addition of a little alkali to increase the conductivity. After electrolizing for ten minutes at a fairly high current density, the solution dissolved gold leaf in about fifteen minutes; this solution also gave a substantial titration with silver nitrate, showing probably that cyanide had formed. Again, treating the original solution with the addition of potassium thiosulphate and boiling, for the purpose of converting any cyanide present into sulphocyanide, on testing this solution with ferric chloride it gave the characteristic blood-red coloration of the iron sulphocyanide—surely ample verification. To make it more confirmatory, I added a solution of ferrous sulphate, to convert any cyanide in the solution into ferrocyanide. On addition of ferric chloride it gave, on boiling, the characteristic prussian blue. I was satisfied that some cyanide was formed by the electrolysis of calcium cyanamide, but as to the amount produced from a given quantity of cyanamide I was in the dark, owing to the peculiar behavior of the titrating solution in presence of organic compounds. It seemed from different experiments made, that the production of cyanide by electrolysis of cyanamide solution, was frequently below three to four-tenths of a pound KCN per ton of solution. During all these tests I used solutions analogous in strength to working solutions. On referring to Ward's "Dictionary of Chemistry," page 314, line 65, Vol. 2, I found that cyanamide combines with potassium cyanate to form mono-potassium amidodicyanate, whose formula is $(CN)_2NH_2OK_2$. Reflecting on this reaction, I thought it probable that, as the cyanide was formed by the electrolysis of the cyanamide, it was oxi-

*Abstract of a paper presented before the American Electro-chemical Society, December 16, 1910.

dized by the current to cyanate, the cyanate then acting upon the unaltered cyanamide to form potassium amidodicyanate. Potassium amidodicyanate when electrolized forms a powerful gold solvent, although it does not give any appreciable reaction on titration with silver nitrate. I have many notebooks filled with interesting data in regard to these reactions, and propose later to publish the results. I may state, however, that when a solution of calcium cyanamide is mixed with a solution of alkaline ferro-cyanide and allowed to stand for a few hours, and even after weeks' standing, it becomes an active solvent for gold without the aid of electrolysis, and when applied to the treatment of ores amenable to straight cyanidation, gives results, equivalent to the simple cyanide treatment, even in very dilute solutions. Further, this process can be used on ores which have already been treated by the cyanide process, that is to say, on residues. There exists in dumps a large proportion of prussian blue, which when treated with alkali becomes in substance, potassium ferro-cyanide and ferri-cyanide, or soluble prussiates.

This opens up a cheap means of treating ores amenable to the ordinary cyanide process as well as ores—residues—which have already been treated by the cyanide solution, that is to say, ores from which the precious metals have not been wholly extracted, and which would not pay for re-treatment by the ordinary cyanide process on account of the cost of cyanide. The following is an example of using this process upon ore amenable to cyanide treatment: Make a solution consisting of 2000 lb. of water containing 1 lb. of calcium cyanamide, 1 lb. of alkaline ferro-cyanide, 1 lb. of lime. The ore is subjected to this solution in the ordinary way, in proportion of 2 parts of solution to 1 part of ore, for a period of say 8 to 10 hr., or until extraction is complete. The following is an example of using the process upon ore which has already been treated by the cyanide process and exposed to atmospheric oxygen which occasions the formation of prussian blue and other ferro-cyanogen compounds. The cre when treated with a solution containing 1 lb. of cyanamide, 1 to 5 lb. of lime (the amount of lime, of course, depending upon the acidity of the ore), in 2000 lb. of water, gave results equivalent to the use of straight cyanide solution when used in the proportion of 2 or 3 parts of solution to 1 part of ore. Again, if the above mixture be electrolized the solution of the gold is extremely rapid. Calcium cyanamide mixed with alkaline sulpho-cyanide, electrolized, immediately becomes cyanide and dissolves gold rapidly. Another amide compound, guanidine carbonate, mixed with potassium cyanate, in conjunction with electrolysis of the solution, dissolves gold leaf in less than ten minutes. I might cite a long list of amidogen compounds in conjunction with cyanate solution if space would permit.

In the midst of this work on cyanogen-bearing materials the thought occurred to me to try the combination of calcium cyanamide and iodine. I forthwith made a solution of calcium cyanamide, adding a solution of iodine, to see if this combination would dis-

solve tellurium, and was rewarded by finding it to be an excellent solvent. I next tried gold leaf with the same combination and had the satisfaction of seeing the gold undergo dissolution in a short time, a scum of calcium carbonate which forms on the surface preventing more rapid dissolution. From this I deduced that the addition of iodine formed cyanamidogen iodide. I am unable to find any reference to this compound in chemical literature which I have read, and I, therefore, take the liberty of calling it by the above name, and further assume the formula to be CN_2I_2. I next tried the action of cyanogen iodide upon a solution of calcium cyanamide, knowing that cyanogen iodine would be destroyed by an excess of alkali, and anticipating that this combination would dissolve telluride of gold. Upon applying the solution to tellerium and telluride of gold, these substances at once dissolved, showing that the dissolution was probably due to cyanamidogen iodide, and I again presume the reaction to be in accordance with the formula

$$2ICN + CaCN_2 = Ca(CN)_2 + I_2CN_2$$

I will now describe the general method of using the Clancy process on the working scale. The ore is crushed by stamps, rolls, ball-mills, or any other efficient form of preliminary grinding mill. The degree of fineness necessary for the process being about 100 mesh, this is probably most economically accomplished by the use of tube-mills. The ore is crushed in a cyanide solution containing calcium cyanamide, sulphocyanide, and the halogen salts. It is, therefore, under the influence of straight cyanide treatment practically after leaving the rock-breakers and until it reaches the agitation tank, where it meets with electrolysis, which acts upon the cyanide solution containing the additional chemicals, and these, under the action of electrolysis, forms powerful solvents for the precious metals, as already explained. Before describing the treatment solution, the manner of dissolving the cyanamide should be known. Calcium cyanamide of commerce comes as a black powder and is about from 58 to 65% soluble in water. It is necessary, therefore, to dissolve the cyanamide in a separate tank and filter it from its insoluble residue. A small air-agitating tank is admirably suited to this purpose, and at the same time it will act as a cyanamide storage tank. The cyanamide solution may be made as highly saturated as desired, and the calculated quantity drawn off when required, and added to the cyanogen-bearing solution. The treatment solution is made up to 2000 lb. water, containing 1 lb. of cyanide, 2 lb. of alkaline sulphocyanide, 2 lb. of calcium cyanamide, and ½ lb. alkaline iodine. If using crushing rolls after the rock-breakers, the product leaving the rolls at about 12 mesh is fed into the tube-mill and converted into pulp, by feeding the mill with the treatment solution and ore, in the proportion of 1 part of ore to 1 part of solution. The discharged pulp, after separation of the oversize, is transferred to the agitation tank to undergo electrical treatment. If it is thought necessary to remove the sulphides before, after, or during the treatment, the following method presents

an ideal scheme. When solution with finely divided ore in suspension contained in the well-known cone-shaped tank in the proportion of 2 of solution to 1 of ore, or in the proportion of 3 of solution to 1 of ore, is agitated, if the agitation be stopped for a few minutes, the finely divided sulphide particles settle to the bottom of the cone, and by simply opening the discharge valve at the cone apex, the sulphide may be completely drawn off together with a small proportion of the non-sulphide pulp. This sulphide product may be run over blankets or some similar contrivance, and the finely divided concentrate collected, the excess of pulp solution being returned to the agitation tank for treatment. Here is a means of eliminating the use of concentrating tables and obtaining a product of high value in small bulk. Again, the pulp being in a finely divided state, the pyrite or sulphide portion is not accompanied with quartz or gangue; therefore, a clean high-grade concentrate is obtained, a result that cannot be accomplished by the use of concentrating tables, without the employment of a large quantity of solution with attendant expenses. The pulp now in the agitating tank is of the correct alkalinity, this being previously established in the tube-mill by adding lime so as to contain from $^1/_{10}$ to $^2/_{10}$ of a pound "protective" alkalinity per ton of solution.

The conductivity of the pulp is adjusted by adding common salt until the required voltage is obtained; 20 lb. of salt per ton of solution will invariably decrease the resistance of the pulp so that the volt motor will register about from 5 to 6 volts. In the majority of cases a current of about 50 amperes per ton of ore is adequate. It will be easily seen that the cost for electrical energy is not by any means prohibitive. With iron-oxide electrodes it is possible to obtain a current density considerably over 50 amperes per square foot of anode surface, so that one electrode 3 ft. long by 3 in. diam. will be sufficient for the treatment of from 3 to 4 tons of ore; in other words, approximately 30 of these iron-oxide electrodes would be required for the treatment of 100 tons of ore per day. If the treatment tank be constructed of iron the tank itself may be used as the cathode. This arrangement would, of course, decrease considerably the cost of installation. The electric generator is the chief item of cost. A low voltage generator, such as a 10-volt machine capable of giving the necessary amperage, can be obtained at any of the electrical warehouses. It is obvious, therefore, that the process may be applied to any existing fine-grinding plant provided with agitating tanks. All that is necessary is simply to introduce the electrodes into the circulating ore pulp containing the necessary chemicals and switch on the current. It is essential in every case to maintain the protective alkalinity at about $^1/_{10}$ of a pound alkali per ton of solution so as to allow of the formation of cyanogen iodide and cyanamidogen iodide. About eight hours' treatment under electrolysis usually is sufficient to obtain the necessary extraction.

After the eight hours' treatment with the current, the pulp solution is brought up to about one pound per ton of protective alka-

linity by adding caustic soda, and the cyanide contents regenerated up to about $^5/_{10}$ to $^6/_{10}$ of a pound cyanide per ton of solution. The regeneration of the cyanide is then accomplished simply by giving the pulp about two hours more current. It will be understood that the reason for adding the extra alkali is, that cyanide regeneration cannot take place in the presence of a halogen compound unless the solution containing sulphocyanides and cyanamide is made alkaline. It will be seen that the whole value of the process depends upon the recovery of the halogen compound. While I have described in the examples a proportion of 2 parts of solution to 1 part of ore, a proportion of 3 parts of solution to 1 part of ore may be used with advantage, that is to say, by using 3 parts of solution to 1 part of ore, a much smaller amount of alkaline haloid may be used per ton of solution; thus giving the same ratio of haloid salt per ton of ore as in the 2 of solution to 1 of ore pulp, and consequently a less proportion of soluble haloid to be displaced by the water wash in the final slime cake.

In this necessarily comparatively brief and incomplete description of the process, the use of chemicals and current have been described, but no mention of costs has been made. I will, therefore, take the following example to represent the typical working solution: 2000 lb. of water containing 1 lb. of cyanide, 2 lb. of sulphocyanide, 2 lb. of calcium cyanamide, and ¼ lb. alkaline iodide. This appears a formidable mixture when looked at cursorily, but on analysis it does not work out beyond the limits of economic treatment. For example:

Lb.	Substance.	Cents.
1	cyanide	18
2	cyanamide	6
2	alkaline sulphocyanide	12
¼	alkaline iodide	35
	Total	71

This would not represent the total cost of one ton of solution, for, notwithstanding the effect of electrolysis, practically all the haloid salt, or salts, previously added, together with the sulphocyanide will be found unimpaired at the end of the operation, the cyanide and cyanamide alone suffering the necessary decomposition. It is clear, therefore, that no matter what the proportion of solution to ore, only the consumption of cyanide and cyanamide per ton of ore is to be taken into account. The amount of cyanide consumption on the ore, in presence of cyanamide, works out at about 1 lb. of cyanide per ton of ore treated. This consumption of cyanide is regenerated at the expense of 3c. for cyanamide, and at the outside 3c. for current (figuring current at 1c. per kilowatt hour), making a total cost of 6c. per ton of ore. Added to the above cost is the cost of the electrical energy necessary for the electrolysis of the ore pulp. The cost of electrical energy for this purpose works out at about 10c. per ton of ore treated, this added to the cost for cyanogen-bearing material and regeneration, would

make a total of 16c. per ton of ore. These figures would represent the total cost, provided that all the solutions were recovered without mechanical loss. From this it is evident that the recovery of the solutions for re-use is a matter of vital importance to this process. The mechanical recovery of the solutions is, therefore, entirely dependent upon the efficiency of the filter employed. The Moore filter eminently fulfils the requirements for the recovery of the solution, inasmuch as it gives perfect uniformity of cake, both in thickness and porosity, which means perfect resistance, and perfect resistance guarantees perfect displacement. In fact, the business combination of the Moore filter with the Clancy process was originally brought about, not from any arbitrary or usual business considerations, but because, after the most rigid examination of all existing filter processes, I learned in my demonstrations that this type of filter was the only one that could be depended on to recover practically all of the solutions. This filter process is too well known to describe minutely. Suffice it to say, when the final step in the cycle of treatment is reached, that is, when the cake has been washed with barren solution, the soluble salts contained in the moisture saturating the cake at this juncture, may be completely recovered by giving the cake the requisite amount of water for displacement. The necessary amount of water for displacement is readily seen without calculation by the diminution of the water level in the wash tank. It has been necessary to mention the unique points of this filter so as to show how the solutions can be recovered with a minimum of mechanical loss and with a minimum of water wash. In a properly constructed Moore filter (type A), the loss of solution under proper manipulation should not exceed 10% of the total solution. It should be understood that the above costs are based on the treatment of rebellious or refractory ores and that they would in all probability be much reduced when dealing with non-rebellious ores.

(February 18, 1911)

The Editor:

Sir—In the *Mining and Scientific Press* of December 31, 1910, there appears an abstract of a paper presented before the Electrochemical Society, December 16, 1910, on "The Clancy Process." In the paper referred to, Mr. Clancy claims as novel the application of calcium cyanamide as a cheap cyaniding agent in the place of the more expensive sodium or potassium cyanide universally employed in the treatment of gold and silver ores by cyanidation. I beg to take issue with Mr. Clancy regarding his claims to originality in this matter. I can show priority in the application of this material to the extraction of gold and silver from ores. Mr. Clancy may claim some novelty, though of doubtful value, in adapting the use of calcium cyanamide to gold and silver extraction by means of electrolysis instead of the more direct method which I have proposed: its transformation into calcium cyanide by subjecting the cyanamide at a red heat to acetylene, water gas, or any hydrocar-

bon gas rich in carbon. This can be effected during the process of manufacturing the calcium cyanamide from calcium carbide. I cannot understand how Mr. Clancy could possibly have presented calcium cyanamide as a cheap and novel substitute of his own invention in place of the more expensive KCN or NaCN for use in the cyanide treatment of ores, when I had already written up its application several years ago.* The essential parts of the patent specifications are quoted below. The whole paper is easily available.

"In the treatment of ordinary gold and silver-bearing ore and tailing by the cyanide process, a cyanide solvent must be used which is not only stable in dilute solutions, such as 0.05 to 0.50% KCN, but must withstand exposure to the atmosphere as well, without decomposition. It must also not contain impurities, such as sulphide, carbide, etc., which would greatly depress the gold and silver-extraction co-efficient of the cyanide solvent. The cyanides ordinarily used, therefore, are the potassium and sodium salts, KCN and NaCN. These are highly refined, snow-white, and almost chemically pure. In this state they are sent to market for precious-metal extraction purposes, and are, necessarily, very costly; from 20 to 25c. per pound. In recent years, however, there has been great development in the cheap production of alkaline-earth cyanide, such as calcium cyanide, $Ca(CN)_2$, and barium cyanide, $Ba(CN)_2$, by means of the electric furnace, from atmospheric nitrogen. I have found that these electric-furnace products, even impure as they are, if they contain 80 to 90% of the theoretical quantity of cyanogen $(CN)_2$, may be adopted and used, in ammonia solutions, as gold and silver solvents. In carrying out my invention, the crude calcium cyanide or barium cyanide is dissolved in ammonia solution of sufficient strength, the insoluble residue solution to settle, and the clear solution applied in the treatment of gold and silver-bearing ore and tailing, not amenable to ordinary cyanide treatment, either on account of excessive cyanide consumption due to cyanicides, or on account of percentage of metals such as copper, nickel, zinc, or cobalt, which render inert the gold or silver solvent powers of a dilute cyanide solution (KCN or NaCN), such as is commonly used, and cause an excessive loss of the cyanide. A working solution, according to my invention, may be made up as follows: Solution of ammonia, 1% or less to 10% (NH_3); calcium cyanide or barium cyanide, 0.05 to 1%. If this solution be too alkaline, a salt of ammonia, such as the sulphate, may be added to counteract such excessive alkalinity. The treatment of the ore or tailing with this solution is carried out in any of the usual known manners, as by leaching or agitating them with such solution; and, where the ore or tailing contains, in addition to the gold and silver, such metals as copper, nickel, zinc, or cobalt, all these metals are simultaneously extracted, and the separation of the gold and silver afterward affected in any ordinary manner. It will be seen that by my

*See United States Patent 911,254, filed September 4, 1907, issued February 2, 1909.

improvement, a very cheap gold and silver solvent is obtained, which overcomes all the objections and conditions above referred to."

Having thus described my invention, what I claim as new and desire to secure by letters patent is: (1) The improvement in treating ore and tailing containing gold and silver, which consists in extracting the precious metals with a solution of ammonia and alkaline-earth cyanide. (2) The improvement in treating copper, nickel, zinc, and cobalt ore and tailing containing gold and silver, which consists in simultaneously extracting all the metals with a solution of ammonia and an alkaline-earth cyanide. Then the following periodicals published articles and abstracts relative to the subject: *Mining and Scientific Press,* May 15, 1909; *Engineering and Mining Journal,* April 17, 1909, February 27, 1909; *Pacific Miner,* March, 1910; *Mexican Mining Journal,* August, 1910; *Chemical and Metallurgical Engineering,* May, 1910; 'Mineral Industry,' Vol. 18, 1909, p. 365. In the above articles, abstracts, and letters, calcium cyanide as a cheap cyanide salt substitute produced from calcium cyanamide as per equation, Ca_2CN_2 plus C equals 2Ca $(CN)_2$, was several times noted and its use recommended by me as a cheap cyaniding agent for extracting gold and silver from their ores.

I trust I have made clear my position in regard to the Clancy process as relates to the use of cyanamide, and the cyanide-engineering profession may judge for itself if Mr. Clancy is entitled to any credit in presenting calcium cyanamide as a novel product for use in the cyanidation of ores. In its intended use, as described in his last paper before the Electro-chemical Society, Mr. Clancy, perhaps unknowingly, but none the less certainly, infringes on my rights and prerogatives in the matter. The fact that Mr. Clancy employs iodide in conjunction with the cyanamide is of no importance to the intrinsic claim of employing cheap cyanamide as a substitute for the expensive cyanides of sodium or potassium. I trust you will publish the above in justice to myself and the cyaniding fraternity.

D. MOSHER.

San Francisco, January 20.

The Editor:

Sir—After a careful perusal of Mr. Mosher's communication, a copy of which I am assuming was addressed to you, as it appears copies have been sent to several leading mining journals, and after careful perusal of his patent specifications and other references which he has made in his communication, I fail to see any cause for discussion as to "priority" in the use of cyanamide in ore treatment. Nowhere, including his patent specifications, does it appear that he suggests the use of cyanamide *per se* as a solvent for the precious metals; on the other hand, he simply proposes, as many others before him have done, the use of calcium cyanamide as a starting point in the manufacture of calcium cyanide, by heating to

a red heat calcium cyanamide with other materials in an electric furnace, and the product of this mixture—calcium cyanide—he proposes to apply to the treatment of ores. The German chemists, Messrs. Frank & Caro, many years ago attempted to produce cyanides by this means, but found that it was not a commercial success. I am reliably informed by those actively interested in the manufacture of cyanamides that fortunes have been spent endeavoring to apply cyanamide in a thoroughly practical and economical way, and they have very generously and frankly conceded that I have solved the problem. It seems to me that it would be more to the point if Mr. Mosher took issue with Frank & Caro for the determination of the question of"priority," for there appears to be little, if any, difference in the result finally accomplished by either of them, since they were apparently working along the same lines and the results quite naturally were the same. It will be apparent to anyone reading Mr. Mosher's specification that he does not use cyanamide in the cyanide solution, but proposes to use his electric-furnace product, which contains calcium cyanide of about 80% cyanogen content prepared by secondary electric-furnace treatment of cyanamide, as already described. In the Clancy process any amide or amidine compound may be used in conjunction with the cyanide solution, such, for example, as urea, or guanidine. The idea of my infringing any rights or prerogatives of Mr. Mosher is amusing. Why does not Mr. Mosher claim priority to an amide or amidine compound in conjunction with the cyanide solution?

I trust that I have made the distinction clear between the Clancy process relating to the use of cyanamide in the cyanide solution, and the use of the calcium cyanide product prepared from calcium cyanamide by special treatment as described by Mr. Mosher. It is clear from reading Mr. Mosher's specification that he does not attempt to claim the use of raw cyanamide in the ore-treatment process, for the simple reason that he was not aware of the fact that cyanamide, amide, or amidine compounds could be used in conjunction with cyanide solution until the publication of my patent specifications and the articles which I have written describing my process.

JOHN COLLINS CLANCY.

New York, February 7.

The Editor: (March 25, 1911)

Sir—I will again have to ask the patience of your readers, as a number of statements in Mr. Clancy's reply to my letter, 'The Clancy Process,' in the February 18 issue of the *Mining nad Scientific Press* demands correction. The following respecting the value and application of calcium cyanamide, or its more valuable equivalent, calcium cyanide, for use in the cyanide and ammonia cyanide treatment of ores will, I trust, enlighten the cyanide-engineering profession. What is of most value to the cyanide operator in the arguments presented both by Mr. Clancy and myself, is the cheapening of

cyanide; to be had at a cost of 3 to 5c. per pound, as against sodium cyanide costing 18c. Mr. Clancy wants cheap cyanide and believes he has it in calcium cyanamide. I want cheap cyanide and feel certain I have it in calcium cyanide. Since I first presented and patented the idea, by all the laws of common-sense, I am entitled to priority in its application in ore treatment. I am certain that the cyanide engineer is already sufficiently overburdened by an ever-increasing number of new mechanical devices and appliances, and to be successful, must these days be pretty well versed in inorganic chemistry. He can scarcely desire to be still further drawn into the more occult domain of organic chemistry as Mr. Clancy is apparently attempting to do.

When Mr. Clancy says that "the Messrs. Frank & Caro many years ago attempted to produce cyanides by this means, but found it was not a commercial success" he is only begging the question; for the process of producing the calcium cyanamide and calcium cyanide, is a mere matter of chemical control, and not of failure on the one hand in producing calcium cyanide and success achieved on the other, in manufacturing calcium cyanamide. Calcium cyanamide, due to its fixed nitrogen being readily convertible into urea and kindred organic nitrogen products, has become of great value as a 'fixed nitrogen,' fertilizer, for which it is chiefly now being manufactured; while the related product, calcium cyanide, would prove poisonous in the soil to plant life, and therefore inapplicable. Even cyanamide works havoc in many instances and must be used with care. For gold and silver extraction work, calcium cyanamide has only half the cyanogen value of that of calcium cyanide, and in the commercial cyanamide, the percentage of cyanogen is so low that its use in cyanidation, even at 3c. per pound, is hardly advisable, as I will also prove. The theoretical percentage of nitrogen in calcium cyanamide, $(NCa)CN$, is 34.22%, while the theoretical percentage of nitrogen in calcium cyanide is 31.5%, the formula being $Ca(CN)_2$. The atomic weights entering into their respective compounds taken from the latest figures are as follows:

	Atomic weights.
Calcium	40.09
Carbon	12.00
Nitrogen	14.01

In calcium cyanamide there is one atom of uncombined nitrogen and one molecule of cyanogen. The free atom of nitrogen uncombined with carbon present in calcium cyanamide, forms, on slaking the product in water, NH_2, or amide, of no known value as a gold and silver solvent. But if this nitrogen atom be united with carbon (which is readily feasible), another molecule of cyanogen is gained or, plainly speaking, real calcium cyanamide is only of half the value, in gold and silver extraction, of c.p. calcium cyanide, due to double the quantity of cyanogen (CN) existing in the latter.

The cyanogen percentage in the various salts that are at present of interest to the cyanide engineer is as follows:

	Cyanogen, per cent.
Sodium cyanide, $NaCN$	53.07
Potassium cyanide, KCN	39.94
Calcium cyanide, $Ca(CN)_2$	56.47
Barium cyanide, $Ba(CN)_2$	28.00
Calcium cyanamide, $(NCa)CN$	32.06
Sodium sulpho-cyanide, $NaCNS$	32.07

A typical analysis of the commercial calcium cyanamide is as follows:

	Per cent.
$CaCN_2$	57.0
Carbon	14.0
Lime	21.0
SiO_2	2.5
Fe_2O_3	4.0
Sulphur, phosphorus, and calcium carbonate	1.5
Total	100.0

Multiplying 34.22% nitrogen present in the real product by 57% the amount of c.p. cyanamide existing in the commercial salt, a percentage of 19.5% nitrogen is obtained. Since only half of this is present as cyanogen, of value in gold and silver extraction, and is equivalent to 9.75% nitrogen, and since this corresponds to 19% cyanogen, it will be seen that even at 3c. per pound calcium cyanamide is by no means a cheap cyanogen product. On an average of a 15% basis of cyanogen contents, calcium cyanamide in comparison with sodium cyanide works out as follows: 1 lb. NaCN 48 to 52% cyanogen is equivalent to 3 1/3 lb. (NCa)CN of 15% cyanogen. At 3c. per pound, 3⅓ lb. (NCa)CN cost 10c., additional freight on 2⅓ lb. probably 2c., total 12c. per pound, as against 19c. for NaCN. With this comparison and the possibility of cheaper sodium cyanide, cyanamide offers for the cyanide operator not much inducement, when the impurities in the calcium cyanamide are taken into account. It is different with calcium cyanide, of a nitrogen content of 19.5%, equivalent to 36.6% cyanogen, and produced at a cost of 4c. per pound. This would make it equivalent to sodium cyanide at 6c., or one-third the present price; at that figure it will certainly prove of value on account of its cheapness, and in ammonia cyanidation of the greatest value due to the fact that in boiling off the ammonia the cyanogen goes along and is practically recovered without loss for continued re-use; or is transformed into ammonia (NH_3) and thus replenishes the ammonia losses in the mill. Calcium cyanamide when dissolved in water forms cyanamide and a host of organic nitrogen products. But returning to first principles and for easy explanation, I will reverse the appellation of cyanamide and call it amide cyanide NH_2CN. This is produced from calcium cyanamide according to an equasion which may be written as below:

$$CaCN_2 + 2H_2O = Ca(OH)_2 + NH_2CN$$

To the cyanide operator amide cyanide will easily compare with ammonium cyanide, NH_4CN, a powerful silver and gold solvent. Instead of calcium cyanamide I will call it calcium-nitrogen

cyanide; it will be seen at once that the nitrogen in calcium cyanamide uncombined with carbon forms amide or amidogen and cannot dissolve gold; it may perhaps silver. The real value of the product is therefore in its cyanogen content, as previously explained. Unless Mr. Clancy has found some special effect in the use of cyanamide in his researches, I can only see a slight commercial value in its adoption in cyanide treatment.

Mr. Clancy claims that he uses cyanamide *per se*. I can not see of what value this can be, even when subjected to electrolysis. It is true that Mr. Clancy protects his cyanide solution through reducing the cyanate by means of the NH_2 group present in cyanamide. The addition of the NH_2 to cyanate and subjecting it to the electric current, by which Mr. Clancy effects its oxidation, forms water and nitrogen, and the cyanate is thus reduced. But, after all, cyanamide is an expensive cyanide product even in its apparent cheapness, and, furthermore, lacks simplicity in its application.

Mr. Clancy forces the suggestion upon me of aiming to manufacture calcium cyanide rather than employ it for cyanide treatment. To this I will reply that I am not directly interested in a process for making calcium cyanide, as that is a matter for manufacturing chemists, but only in applying the product to the treatment of ores.

Mr. Clancy, in his letter, says that I propose to use an electric furnace product, which contains calcium cyanide of about 80% cyanogen content, prepared by secondary electric furnace treatment of cyanamide. This is contrary to the facts. In the *Electrochemical and Metallurgical Industry,* July, 1909, in an abstract, the following is stated regarding the manufacture of calcium cyanamide, and it will be apparent to anyone that it is an electric-furnace process, pure and simple. "The furnaces have hence to be heated; but if the heat were applied from outside, the temperature would rise too high on the walls and decompose the formed cyanamide. The heating is, therefore, effected in the mass itself in heat-insulated furnaces by means of electrically heated rods of carbon; after a while the heating can be interrupted and will proceed automatically, all the more readily the purer the carbide."

In conclusion, I will say that I am the last one who would attempt to discourage or disparage Mr. Clancy; for he certainly has done a great deal of research work in cyano-metallurgy, and by all means is entitled to the support of the cyaniding fraternity. But, at times, Mr. Clancy forgets to give credit to others who are also working on similar lines, and in his proposal to apply ferric oxide electrodes for use in cyanide treatment does not even mention the original inventor or patentee of these iron oxide electrodes, H. Speckter, U. S. patent No. 931.513, August 17, 1909. The patent is assigned to the Gresheim Electrode Co., of Gresheim, Germany. These electrodes are intended for use in the electrolysis of salt for the production of chlorine and caustic soda.

D. MOSHER.

San Francisco, March 14.

The Editor: (April 29, 1911)

Sir—It will be seen from reading D. Mosher's article on the Clancy Process published in your issue of March 25 that there is no ground for the comments and figures which he has given. If Mr. Mosher wants cheap cyanide and thinks he has it in calcium cyanide, it has nothing to do with my process. I do not use calcium cyanide nor do I attempt to produce calcium cyanide from cyanamide. In all cyanide plants using lime, surely a certain amount of calcium cyanide may be formed. Would Mr. Mosher then attempt to restrict the use of lime in the cyanide process unless he is paid a royalty? That Mr. Mosher has shown in his writings that he does not use cyanamide or calcium cyanamide, may be seen from the disparaging manner in which he speaks of my use of cyanamide in conjunction with the cyanide solution. He states that he cannot see any value in the use of cyanamide other than protecting the cyanide through reducing the cyanate by means of the NH_2 group present in the cyanamide, and forgets that when I use oxidation means other than the atmospheric oxygen, a considerable proportion of cyanate may be formed in the solutions. If this destruction of cyanide to cyanate can be prevented by the use of cheap cyanamide in presence of sulphocyanides is it not a consummation devoutly to be desired? To demonstrate that Mr. Mosher has not caught even a glimpse of the claims of my process, suffice it to insert a quotation from his article and leave the rest with the readers: "Unless Mr. Clancy has found some special effect in the use of cyanamide in his reseaches, I can only see a slight commercial value in its adoption in cyanide treatment."

New York, April 4. JOHN COLLINS CLANCY.

CYANIDE REGENERATION

By B. George Nicoll

(March 16, 1912)

*With the introduction of the cyanide process it appeared that finality in gold-extraction processes had nearly been achieved, but the early application of the process to many ores was accompanied by such unsatisfactory results that eventually only those which contained no refractory elements were selected for cyanidation. In accumulated dumps the difficulty lies chiefly in the high consumption of cyanide, owing to the fact that other metals than gold generally go into solution readily, and the baser metals have sometimes to be fairly well leached out before the gold is attacked. This is particularly the case where copper is present in small quantities, and in the cyanidation of auro-cupriferous ores many variations of the cyanide process have been tried, mainly aiming at effecting a reduction in the chemical consumption. No plant operating on auro-cupriferous ores has yet overcome copper troubles, and so long as copper is readily soluble in a cyanide solution, in no such case will the initial consumption of cyanide be very materially re-

*Abstract from *The Mining and Engineering Review.*

duced. The presence of copper in solution gives rise to precipitation difficulties, and these have so effectually resisted correction that precipitation on zinc-shavings had to be abandoned in favor of other methods. In the early operation of the process the only other form of precipitation practised was the Siemens & Halske electrical process, with precipitation boxes about 4 by 8 by 30 ft., consisting of about six compartments, each 4 by 8 by 5 ft., and each compartment containing anodes of iron enclosed in hessian cloth and cathodes of lead foil. The distance between anodes and cathodes was about 6 in., and a current density of 0.02 amperes per square foot of anode surface was generally employed. As low as this current density was, it was sufficient to cause the anodes to decompose rapidly, with consequent formation of ferro-cyanides and ultimate loss of cyanide during the passage of the solutions through the precipitation plant. The only advantage that the process offered over zinc precipitation was that more metal was recovered, and the solutions were returned to the vats in a fairly clean state. The chief difficulty, the high consumption of cyanide, had not been remedied; on the contrary, the cyanide consumption had slightly increased, and the advantages obtained scarcely compensated for the increase.

In 1903 a great improvement in electrical precipitation was effected by Charles Butters, of California. Butters used anodes of lead sheets peroxidized by electrolysis in a solution of permanganate of potassium, and sulphuric acid. These anodes resisted oxidation to a much greater degree than iron anodes, and withstood a current density of 0.5 amperes per square foot without apparent decomposition, and, owing to the use of a greater current density, the size of the precipitation plant was greatly reduced. Butters erected at Virginia City an electrical precipitation plant consisting of boxes 29 ft. long, 10 ft. wide, and 4 ft. deep, and with V-shaped bottoms. Each box had 12 compartments, and each compartment contained 18 anodes and 17 cathodes, spaced 1 in. apart. The anodes consisted of 0.185-in. lead plates, covered with lead peroxide, and not enclosed in any way. The cathodes were plates of ordinary commercial tinned steel, and fitted with an iron strip along the tops to keep them rigid and to connect them together. In using such a high-current density, the gold is precipitated as a pulverulent deposit on the tin, and is periodically wiped off, falling to the bottom of the box, where it remains until cleaned up. In the operation of the Butters plant a regeneration of cyanide was observed, and this was increased as the decomposition of anodes was reduced. It also increased with the increased precipitation of copper and the consequent liberation of cyanogen, which, in the presence of excessive alkali, reformed active gold solvents. It was not however, until this plant had been in operation for some time that the possibility of the recovery of much cyanide was fully appreciated, and, by carefully regulating current density, alkalinity, and other conditions, Butters was able to regenerate sufficient cyanide to defray the cost of running the precipitation plant.

In September 1905 a patent, entitled 'Method for recovering precious metals from solution,' was granted to Isaac Anderson and Michael Scanlon, metallurgical chemists, of Arizona. In this process it was claimed that the silver and gold could be precipitated in such a manner that the solution could be readily restored to its original activity with but little loss of the original solution and at a small cost for regeneration. This process is based upon the fact that when sulphuric acid is added to a cyanide solution of the precious metals in which sulphides and chlorides are present, the sulphuric acid so added will combine with the potassium of the potassium cyanide to decompose the double cyanides formed by the action of the potassium cyanide on the precious metals, and the metals freed from combination with the potassium cyanide will then combine the sulphur and chlorine present to form insoluble sulphides and chlorides, which will be precipitated, leaving the cyanogen in the liquid in the form of hydrocyanic acid, or in combination with the radicals with which the sulphur and chloride have been combined. The precious metals having been precipitated as chlorides or sulphides, the superabundant liquor can be decanted off, and the potassium cyanide may be regenerated by decomposing the potassium sulphate formed by the action of the sulphuric acid from the solution. The decomposition of the potassium sulphate and the removal of the sulphuric acid may be readily accomplished by introducing lime into the solution and decanting off the precipitated gold and silver, the lime so added causing the decomposition of the potassium sulphate with the formation of insoluble calcium sulphate and simultaneous regeneration of potassium cyanide. In the operation of this process it was claimed that a regeneration of 90% of cyanide could be effected. At the time of the introduction of this process there was little demand for the advantages which it offered, and there is no record of its operation on a working scale, except on the works controlled by the inventor, where, apparently it was satisfactory.

In 1908 W. H. Wheelock, an American chemist, in experimenting with the action of sulphuric acid on cyanide solutions, practically rediscovered the process previously operated by Anderson, and contributed an article to the *Mining and Scientific Press* (December 18, 1909) setting forth the results of his experiments. After perusing Wheelock's article I then conducted a number of experiments at the works of Mitchell's Creek Gold Recovery Co., Bodangora, N. S. W., on the lines laid down by Wheelock, and the following are the particulars of one of these tests: 160 lb. KCN solution containing 0.23% free cyanide, 1.5 lb. lime, and about 5 lb. copper per ton of solution was taken from the tops of the precipitation-boxes and put into a clean cyanide case for treatment. Sulphuric acid was then added (the solution being gently stirred during the addition of the acid) till a precipitate ceased to form. It took 2 lb. of acid to complete the reaction. After standing for two hours, a solution from 5 lb. of lime was added (the portion of lime that did not dissolve in water being discarded); lime solution was added

only until the acidity was neutralized. After stirring and allowing the precipitate to settle, the solution was again tested for KCN and it was found to have risen to 0.695% or a gain in KCN of 10 lb. per ton of solution.

These results indicate that under careful manipulation the process of precipitation by sulphuric acid and regeneration of the cyanide is commercially possible when the following general conditions are observed: The solution to be treated should contain very little free cyanide, because if free cyanide be present a fair amount of acid will be consumed in neutralizing it before precipitation actually commences. This cyanide would be totally lost, and so would the acid used to neutralize it. The solution should be as nearly saturated with copper as possible; this can be obtained by passing the solution through successive vats until it will pick up no more metal. Acid should be added until the further addition of acid ceases to make the solution turbid. An excess of acid will prevent a good regeneration and cause a subsequently greater consumption of lime to restore alkalinity and regenerate the KCN. The precipitate formed on the addition of acid should be removed as quickly as possible from the solution and lime added before much HCN could have had time to escape.

Anderson claims that a regeneration of over 80% of the original cyanide can be obtained in the treatment of solutions carrying 0.5% of copper, and that the precipitate will contain 50% of copper; but, of course, the results must depend upon other conditions than only the amount of copper present.

At the Mitchells Creek works several hundred tons of solution was profitably treated by acid regeneration, but lack of vat accommodation, the large amount of precipitate to handle, and the absence of conveniences to systematically continue this method placed it at a disadvantage when compared to improved electrical precipitation. The latter plant will operate continuously with very little attention, and with only a small power-cost; while facilities for acid regeneration would require large storage capacity for solutions and a special smelting plant to deal with the precipitate. On the same works an experimental electrical precipitation plant on the Butters principle was installed, and this plant consisted of a box 10 ft. long by 12 in. wide and 12 in. deep at the sides, and tapering to 18 in. deep in the middle and containing eight compartments. About 120 sq. ft. of cathode surface was employed, and from 3 to 8 tons of solution passed through the box in 24 hours. The current density rate of flow and other conditions were occasionally varied, but the average of daily returns of this plant during about three months' run is given below:

Rate of Flow: Tons per 24 hr., 3.5.	Top of box.	Tail of box.
Au (dwt.)	1.6	0.4
Cu (lb. per ton)	4.68	3.83
KCN (%)	0.05	0.125

Current Density: Amperes per sq. ft. of anode surface, 0.5.

This average is from the total returns recorded. During the trials two samples were taken daily, each sample being from a continuous siphon, and included in the returns are a number of samples taken while the current was off, and during which period a loss of cyanide was recorded. As a result of these experiments a large electrical precipitation plant was installed, but this has not been long enough in operation to furnish any valuable data. Since this plant started it has effected a regeneration of from ½ to 2 lb. KCN per ton of solution treated, but better results than these are expected when the most suitable condition of the soluions for precipitation has been determined.

Since the installation of the above plant a patent has been granted for a method of electrical precipitation wherein rods of Acheson graphite, each ½ in. by 24 in., are placed side by side to form a plate, and cathodes of corrugated or waterproofed felt or cardboard, which has been made superficially electrically conductive by treatment with an adhesive compound and Acheson graphite powder. Acheson graphite, by reason of its purity and high conductivity, will withstand a very high current density as an anode without injury, and the fine graphite is found to impart a highly conductive surface to the treated cardboard. Cathodes treated in this way cost about £10 per 5000 sq. ft., and this area is sufficient to obtain good precipitation from 200 tons of solution per 24 hours. The arrangement of electrodes in this plant is such that the greatest possible number of AuCN and CuCN molecules will be brought into contact with the cathodes during the passage of the solutions through the box. In electrical precipitation more depends on diffusion of the molecules than on variations of other conditions, and increasing the cathode area increases the percentage of precipitation more than does increased current density, as decomposition of the metal bearing molecules only occurs when they actually make contact with the cathode. By the use of anodes, consisting of round rods placed side by side to form a plate, and cathodes of finely corrugated, electrically conductive cardboard or waterproof felt, the area of electrode surface in a box of given size is more than 100% greater, and the solutions in passing between the uneven surfaces of the electrodes will be more diffused and less liable to track than is the case where the electrodes are ordinary flat sheets. The cathodes require to be made sufficiently strong to prevent warping, and this can be done by running a wire around the sides and bottom, the ends of the wire being bent over so as to dip into the mercury groves to make electrical contact. An additional advantage in having the cathodes corrugated is that the precipitated metal will not peel off and touch the anodes, causing short circuits. If the cathodes are quite rigid the distance between anodes and cathodes can be reduced to about ⅜ in., the clearance required being only enough to prevent the accumulation of precipitate between the plates. The rods composing the anodes are fixed together firmly. Kerosene is poured into the box to form a layer

about ½ in. deep on the top of the solution, to prevent the accumulation of froth and the escape of hydrocyanic acid.

Precipitation and subsequent regeneration of KCN depends upon the fact that when the metal-bearing molecules in their passage through the box come in contact with the cathode they are immediately decomposed, the metal adhering to the cathode or falling to the bottom of the box, where it remains until the clean-up. The energy imparted to the liberated cyanogen is expended partly in forming compounds with free alkali, and partly in the decomposition of water and formation of hydrocyanic acid. Where the anode used has not a high resistance to oxidation, most of the liberated cyanogen would be lost in the formation of compounds with the anode material.

The total cost of a plant to treat 300 tons of solution per 24 hours would be about £500, the attention required would be negligible, and about 12 b.hp. would be required to furnish the necessary current. A small recovery in cyanide and the higher extraction possible from cleaned solutions would soon return the cost of the plant. Attendance, power costs, and cathode renewals would not be as great as the cost of attendance, zinc, etc., in a zinc precipitation plant. In the former plant the copper also would be recovered.

In some cases it would appear to be an advantage to use a combination of zinc and electrical precipitation, the solutions passing first through the zinc-boxes, where the gold would be saved, the function of the electrical boxes being to purify and regenerate the solutions before returning them to the vats. On account of the difficulty in getting complete precipitation of the metals without excessive cathode surface in an electrical precipitation plant, a judicious combination of zinc and electrical precipitation in the cyanidation of auro-cupriferous ores is almost certain to show greater economy than either method alone.

The latest process claiming to effect a saving in cyanide and a good extraction from raw sulphide or sulpho-telluride ores has recently been patented by J. C. Clancy and is now in operation at the 100-ton plant of the Ajax Gold Mining Co. at Victor, Colorado. This process depends upon the constant formation by electrolytic action of cyanogen iodide in the pulp during agitation. Mr. Clancy claims to have reduced the consumption of cyanide to mechanical loss only. I have conducted a number of experiments in accordance with Clancy's directions, but up to the present time the results have not been uniform, and trials of the process on a larger scale are now being carried out. In Australian mining at the present time more depends upon the discovery of some process which will profitably extract the metals from poor sulphide ores than on the discovery of new mining fields, and the success of the Clancy process is greatly to be desired. Co-operation between metallurgists exploiting the electro-chemical processes would porbably bring success where isolated experiments are certain to fail.

(April 6, 1912)

The Editor:

Sir—In the article on 'Cyanide Regeneration' by George Nicholl in the *Mining and Scientific Press,* March 16, 1912, there are some statements, apparently quoted by Mr. Nicholl, which I believe are open to discussion.

Descriptive of the process patented by Isaac Anderson and Michael Scanlon, it is stated that, "This process is based upon the fact that when sulphuric acid is added to a cyanide solution of the precious metals *in which sulphides and chlorides are present,* the sulphuric acid so added will combine with the potassium of the potassium cyanide to decompose the double cyanides formed by the action of the potassium cyanide on the precious metals, and *the metals freed from combination with the potassium cyanide will then combine the sulphur and chlorine present to form insoluble sulphides and chlorides, which will be precipitated,* leaving the cyanogen in the liquid in the form of hydrocyanic acid, or *in combination with the radicals with which the sulphur and chlorine have been combined. The precious metals having been precipitated as chlorides or sulphides,* the superabundant liquor can be decanted off, and the potassium cyanide may be regenerated by decomposing the potassium sulphate formed by the action of the sulphuric acid from the solution. The decomposition of the potassium sulphate and the removal of the sulphuric acid may be readily accomplished by introducing lime into the solution and *decanting off the precipitated gold and silver,* the lime so added causing the decomposition of the potassium sulphate with the formation of insoluble calcium sulphate and simultaneous regeneration of potassium cyanide. In the operation of this process, it was claimed *a regeneration of 90% of cyanide could be effected."* The italics are mine.

Two or three years ago I made some experiments upon cyanide regeneration with the use of sulphuric acid, and, as Mr. Nicholls states, practically rediscovered the essential features of the process accredited to Messrs. Anderson and Scanlon, as, at that time, I was unaware that any previous experiments had been made along the same lines. Consequently I am quite interested in any articles upon the subject, and usually regard them rather critically. I realize, however, that the practical value of the process in comparison with processes of wider adaptability and greater proved utility, may not justify such a lengthy contribution.

Certain conclusions which I based upon the experiments mentioned are somewhat at variance with statements made in the paragraph quoted. For instance, I do not believe that the presence of sulphides and chlorides is necessary to effect the precipitation of the heavy metals from a cyanide solution when sulphuric acid is added. If the sulphides and chlorides of the heavy metals are to be inferred, they would hardly remain as such in the cyanide solution, but would enter into chemical reaction therewith, possibly as follows:

$$2CuCl + 2KCN = Cu_2(CN)_2 + 2KCl$$

$$\text{or, } 2CuCl + 4KCN = K_2Cu_2(CN)_4 + 2KCl$$

In the presence of chlorides or sulphides of the alkali metals, the ultimate result, at least, of the reaction occurring upon the addition of sulphuric acid, I believe, would be the same as though they were not present, presumably

$$K_2Cu_2(CN)_4 + H_2SO_4 = K_2SO_4 + Cu_2(CN)_2 + 2HCN$$

This brings us to the statement regarding the precipitated metals and their chemical form. The results of my experiments, the equasion just cited, and other data led to the conclusion that they are cyanides; in the case of copper, probably $Cu_2(CN)_2$. In support of this conclusion, I made a number of tests upon the precipitate by the process, treating a known quantity with acid in a stoppered flask. A tube leading from this flask conveyed the gasses arising upon digesting the precipitate in acid, through a beaker of NaOH solution. Upon the conclusion of this treatment, titrations of the NaOH solution with silver nitrate showed sufficient syanogen present, after making allowances for experimental error, to indicate that the reaction is as cited.

It further appeared that practically all of the cyanogen released remained in the solution in the form of hydrocyanic acid, rather than in any more stable form, as the odor of this acid was noted and, upon agitation or long standing, the available cyanogen content decreased, until, in time, it was practically nil, the decrease being in direct ratio to the amount of agitation or length of time the acidified solution remained exposed. Nor would the subsequent addition of acid to break up any cyanogen compounds which may have formed serve to restore any of the original strength of solution available in cyanogen. It therefore seemed fair to surmise that practically all of the released cyanogen remained in the solution in a volatile form.

In my experiments it was noted that an almost complete removal of the gold and silver from the solution was effected with the precipitation of the baser metals, and assays of the precipitate showed the precious metals present therein in the quantities and proportions one would expect in that case. No attempt was made to isolate them in the form thrown down from the balance of the precipitate or to determine in what chemical form they are precipitated. As they are soluble in cyanide solution in a manner similar to the baser metals, it is not unreasonable to infer that the reactions which cause their precipitation and the resultant precipitate are analogous to those of the baser metals. In regard to the statement that the precious metals are precipitated as chlorides or sulphides, one would suspect a misquotation.

From the paragraph quoted, I gather that lime is added before the precipitate produced by the acid is removed. In that case, as my experiments indicated, the lime would neutralize the acid present and restore the hydrocyanic acid to a stable and available form, either as calcium cyanide, or, more probably, as potassium cyanide, by breaking up the potassium sulphate as described. This potassium cyanide, in turn, would re-dissolve the precipitate wholly

or in part, dependent upon the length of time of contact. One would merely have moved in a circle, with the net result of the loss of some available cyanide, due to the escape of hydrocyanic acid during the process, and the loss of the acid and lime used. I found it wise to effect the removal of the precipitate as completely as possible before the treatment with lime. It is further stated that a "regeneration of 90% of cyanide could be effected." If the general equation of the reaction cited is correct, the possible theoretical saving, based upon the cyanogen in the solution, not that originally used, is not over 50%, unless the precipitate, which contains the remainder, is treated.

In conclusion, permit me to state that the deductions here given are based primarily upon my individual experiments, aided by a limited bibliography dealing in an unsatisfactory manner with the reactions occurring in cyanide solutions, and, furthermore, commercial practicability rather the requirements of chemical research dictated the methods employed, consequently I may have overlooked some salient points which might be obvious to others better informed upon the subject. I am sending this contribution in the hope that, if such is the case, criticisms will be forthcoming. I might add that I was granted a patent upon the process as I worked it out, application for the same having been made before I learned of the experiments performed by Messrs. Anderson and Scanlon, consequently there may be some essential undetected differences in our methods.

R. P. WHEELOCK.

Kingman, Arizona, March 22.

REGENERATION OF CYANIDE SOLUTION

By W. D. Williamson

(July 13, 1912)

In 1899 while treating cupriferous tailing in North Queensland, the excessive cyanide consumption, due to the presence of copper carbonates, made profitable treatment exceedingly difficult. Solutions rapidly fouled, precipitation was very unsatisfactory, and extraction decreased until the only alternative was to discard the fouled solution and make up fresh solution. Changes of solution were very frequent and involved considerable expense and loss of time. This trouble was finally overcome by preliminary leaching with sulphuric acid, about 12 lb. of acid being used per ton of tailing treated. Later, when sulphide ore had to be treated, sulphuric acid leaching was useless.

Experiments made on solutions, which had become charged with copper, demonstrated the fact that a very considerable saving of the combined cyanide could be effected by the addition of sulphuric acid to the solution, in sufficient quantity to precipitate all the copper as $Cu_2(CN)_2$—namely:

$$Cu_2(CN)_2 2KCN + H_2SO_4 = Cu_2(CN)_2 + K_2SO_4 + 2HCN$$

The insoluble $Cu_2(CN)_2$ was separated by filtration, and the clear filtrate containing the HCN neutralized with a slight excess of alkali.

The experiments were sufficiently encouraging to induce me to continue the method on a working scale. As solutions became overcharged with dissolved copper, sulphuric acid was added to the sumps in sufficient quantity to complete the above reaction. The precipitated $Cu_2(CN)_2$ was settled, and the clear supernatant liquor was drawn off and neutralized in a separate sump, and was then used as a fresh clean cyanide solution. The settled $Cu_2(CN)_2$ was collected and saved for shipment to smelter. The possible recovery of the cyanogen in combination with the copper as $Cu_2(CN)_2$ still gave me material for experiment.

Heating with acid I consider impracticable, as it would involve distillation of the liberated HCN, and probably excessive loss of cyanogen, owing to the decomposition of the cyanogen radical in the presence of heat and strong mineral acids. Eventually I found that an almost complete recovery of the cyanogen in $Cu_2(CN)_2$ could be obtained by suspending the $Cu_2(CN)_2$ in water, and passing sulphuretted hydrogen when the following reaction takes place:

$$Cu_2(CN)_2 + H_2S = Cu_2S + 2HCN$$

The insoluble Cu_2S was separated by filtration and the clear filtrate containing HCN neutralized with alkali.

Finally, I combined the two operations by adding to the cupriferous cyanide solution sufficient Na_2S, to precipitate all the copper present, afterward adding the requisite amount of sulphuric acid to complete the following reaction:

$$2KCN.Cu_2(CN)_2 + Na_2S + 2H_2SO_4 = Cu_2S + 4HCN + Na_2SO_4 + K_2SO_4$$

The precipitated Cu_2S carries down with it the precious metals contained in the solution and is collected, dried, and shipped to smelter. The clear liquor is decanted or filtered, and sufficient alkali added to combine with the free HCN liberated by the previous treatment.

This process, which theoretically should recover all the cyanogen in combination in cupriferous cyanide solutions, has never been used to my knowledge on a working scale, but in laboratory experiments as high as 90% of the combined cyanogen has been regenerated and made available as a clean copper-free cyanide solution.

Commenting on this method, Andrew F. Crosse said: "Some years ago I was engaged in investigations in the cyanide treatment of cupriferous ore near Pilgrim's Rest. During my experiments I found that the solutions were always highly charged with copper, and I was unaware that anyone else had the idea of treating cupriferous cyanide solutions with sulphuric acid. I made many experiments. It was difficult at first to find out exactly the right amount of acid required. Then I made a long series of experiments to find out how much gold and how much silver was precipitated. I never

obtained 100% of gold by precipitation; I used to get, if I remember rightly, 88 to 97%. I think that nearly all the silver was precipitated, but not quite all the copper. It was a very nice precipitate to handle; it would settle very easily. I treated over a ton of solution in this way. I then regenerated the hydrocyanic acid solution, after decantation, by means of caustic soda.

"The second part of Mr. Williamson's paper is certainly an improvement on what I did. I simply took the precipitate, dried it, calcined it, and melted it, and I got a little bar containing about 95% of copper, 1% of gold, and 2% of silver; but I certainly think that where sulphuric acid can be obtained fairly cheaply, and where a certain amount of cyanide is decomposed by copper in the ore, the method is worth considering. Supposing some gold does remain in solution, the solution is simply passed through the zinc-box after regenerating with caustic soda.—*Journal* Chem., Met. & Min. Soc. of South Africa.

POTASSIUM CYANIDE

(March 4, 1911)

The Editor:

Sir—It was refreshing to see, on the front cover of a recent issue of your paper, that a firm of chemical manufacturers is attempting to establish a more sensible standard for cyanides, and it is to be hoped that engineers and the press will fall in line. The method of expressing the percentage of all cyanides in terms of potassium cyanide is crude in the extreme, to say nothing of the absurd appearance of a content of considerably over 100% that it gives to a number of cyanides.

While on the subject of cyanide, I should like to raise a protest against the use of the symbol Cy, that is so often found in some of the best text-books and in the writings of some of our foremost metallurgists, instead of the proper symbol CN. The same applies to a number of other substances, as Am for NH_4, and sometimes Aq for H_2O. So long as the radical goes in and out of combination in entirety it does not matter so much, but when the chemical action is such that the radical is broken up, these supposed abbreviations, a slovenly form of writing, at best, give to an equation expressing the reaction a one-sided appearance. Note, for instance, when abbreviations are used, the mixed-up appearance of the equation expressing the reaction that takes place when concentrated sulphuric acid is added to crystallized potassium ferrocyanide:

$$K_4FeCy_6 + 6H_2SO_4 + 6H_2O = 2K_2SO_4 + FeSO_4 + 3Am_2SO_4 + 6CO$$

and the clearness when the proper symbols are employed:

$$K_4Fe(CN)_6 + 6H_2SO_4 + 6H_2O = 2K_2SO_4 + FeSO_4 + 3(NH_4)_2SO_4 + 6CO$$

Fortunately, Aq is seldom used for H_2O, but if I had used it in the first equation it would have made it even more complex.

Your paper and its sister contemporary, *The Mining Magazine,* have been making an excellent fight to bring about the standardization of technical terms, and have accomplished much. Surely these cases I have instanced require to be amended.

F. H. MASON.

San Diego, California, January 19.

SILVER IN SULPHOCYANATE DETERMINATIONS

By E. M. Hamilton

(March 11, 1911)

The following notes deal with facts that are no doubt well known in general chemistry, but their relation to some of the standard methods recommended for analysis of cyanide plant solutions is apt to be overlooked; therefore, a short account of a recent personal experience may be useful to others, especially since most writers on the chemistry of cyanide solutions have not mentioned these points.

In the course of an examination of some solution resulting from the treatment of concentrate, I used the method most commonly given for the determination of sulphocyanate; that of titrating with permanganate after removal of the ferrocyanide by precipitation as prussian blue. The solution contained 0.5% free cyanide, 0.1% free alkali (in terms of CaO), no zinc, and about 80 oz. of silver per ton. Acidifying with sulphuric acid produced a heavy precipitate of silver which was filtered out before proceeding farther. On addition of ferric chloride to the filtrate, no prussian blue was formed, and there was no indication of sulphocyanate. The operation was then repeated, using hyprochloric acid instead of sulphuric, with the result that 0.1% KCNS was indicated on titration with permanganate, but prussian blue remained absent.

After trying a number of experiments with made-up solutions I concluded that in the case of the sulphuric acid the sulphocyanate was taken up by the silver and filtered out in the precipitate. To test this, 50 c.c. of the made-up solution, containing 0.044% KCNS, was acidified with H_2SO_4 and filtered; the precipitate and filter paper were well washed with hot water and the washings added to the filtrate, which was then titrated and gave 0.001% KCNS. The precipitate was redissolved in 100 c.c. of 0.5% cyanide solution, agitated with zinc-dust, filtered, the filtrate acidified with H_2SO_4 and titrated, giving 0.0432% KCNS. This, added to what was found in the filtrate, made a total of 0.0442% KCNS, and showed that the deficiency (amounting to almost the whole of the KCNS originally present) was held in the silver precipitate. The experiment was repeated, using HCl instead of H_2SO_4, with the result that there was obtained in the filtrate 0.042%; obtained by redis-

solving precipitate 0.002%; or a total of 0.044% KCNS. It would thus seem that the results of acidifying with sulphuric acid are only produced to a very slight extent with hydrochloric.

Following this, experiments were made by adding previously prepared and well washed precipitates of AgCN and AgCl separately to a solution of KCNS in distilled water. In the case of AgCN only a trace of sulphocyanate was found in the filtrate while the parallel test with AgCl showed 0.011% KCNS out of a total of 0.0537%. It was thus evident that silver in both these forms would combine with sulphocyanate in solution, and the question arose, why was this result apparent to such a slight extent in the original tests on the made-up solution when HCl was used for acidification? The only difference seemed to lie in the presence or absence of free HCl, so to the previously prepared silver chloride precipitate a little HCl was added before agitating with the KCNS solution (0.049% by titration). The filtrate from the above agitation gave 0.05% KCNS, and showed that the presence of free HCl had prevented the AgCl from reacting with the sulphocyanate.

The conclusions to be drawn would seem to be: (1) When acidifying a cyanide solution containing silver with sulphuric acid, preparatory to titration with permanganate for potassium sulphocyanate, the silver reacts with any KCNS present, converting it into sulphocyanate of silver, which is filtered out and thus removed from the sphere of titration. (2) Although silver chloride will also react with KCNS to form AgCNS, yet this interference is almost entirely prevented by having free HCl present, so that if the permanganate method be chosen practically accurate results may be obtained by using hydrochloric acid instead of sulphuric.* (3) Correct results may be obtained with sulphuric acid after first removing the silver from the solution by precipitation with zinc-dust.

In regard to the latter method, however, there is a point to be observed. When ferrocyanide is present and it is desired to estimate this by separation as prussian blue, the zinc taken up in the precipitation will throw down the ferrocyanogen as zinc ferrocyanide, on acidifying the solution, so that on addition of a ferric salt no prussian blue will be formed. This result can probably be minimized by pouring the solution to be tested into a previously acidified solution of ferric salt, as explained by Clennell ('Chemistry of Cyanide Solutions'). A similar reaction takes place, only in a less degree, in the presence of silver, for on taking a made-up solution containing about 80 oz. of silver per ton, 0.1% KCNS, 0.04% $K_4Fe(CN)_6$, 0.5% free cyanide, and 0.1% CaO, and precipitating the silver with HCl, not a trace of prussian blue was formed on addition of ferric chloride to the filtrate. When the percentage of $K_4Fe(CN)_6$ in this solution was increased above 0.04% and the operation repeated, some prussian blue was thrown down. For sulphocyanate determinations alone, I have found the permangate method gives concordant results, and

*Sutton, Volumetric Analysis, 'Precautions in titrating with permanganate', points out that if HCl be used it should be very dilute to avoid vitiating the result by liberation of chlorine.

when ferrocyanide is present it appears to make no difference whether this is eliminated as prussian blue, or as zinc or silver ferrocyanide, provided that the precipitate and filter paper are thoroughly washed. But if, at the same time, indications of ferrocyanides are being watched for, and if silver is present or if zinc-dust has been used to remove the silver, the absence of prussian blue in the filtrate on adding a ferric salt cannot be taken as evidence that there was no $K_4Fe(CN)_6$ originally present in that solution.

To sum up, it would appear that the simplest procedure for determining sulphocyanate in presence of silver is by acidifying with hydrochloric acid. It should be first ascertained by preliminary trial about how many cubic centimetres of HCl are needed to precipitate the silver and to leave a small excess of acid to act as a protective for the KCNS. Then the necessary quantity should be placed in a beaker and a measured amount of the solution to be tested added to it, and the silver precipitate filtered out. An aliquot portion of the filtrate may then be taken, and ferrocyanide (if still present) removed by addition of ferric chloride, care being taken that the precipitate and filter paper are thoroughly washed. The filtrate and washings should then be further acidified, if not already sufficiently acid for purposes of the titration (avoiding an unnecessary excess), and the permanganate test proceeded with in the usual way.

(July 8, 1911)

The Editor:

Sir—The publication of the interesting article by E. M. Hamilton, dealing with the sulphocyanate determinations in the presence of silver, prompts me to send you extracts from some notes that I made at the time of starting up a high-grade silver-ore cyanide plant in Mexico some little time ago. Daily titrations showed that the solution entering the zinc-boxes was invariably 30% lower in total cyanide than the effluent. Free cyanide titrations, on the other hand, showed no difference at either end.

The explanation was simple, and I accounted for the difference by the fact that the incoming solution carried large quantities of silver in the form of the double cyanide of silver and sodium. The cyanide in this compound would not appear in an ordinary estimation for total cyanide made with the aid of alkaline iodide indicator. During passage through the boxes, however, the silver was liberated and the double cyanide of zinc and sodium formed. The available cyanide in the latter could then be titrated for by ordinary means. The absence of copper made it possible, by a comparison of total cyanide titrations of incoming and outgoing solutions, to estimate, with fair accuracy, the amount of silver which was being deposited in the boxes.

A. W. ALLEN.

Uruguay, South America, May 25.

PRELIMINARY TREATMENT OF WATER AND AIR IN CYANIDE PROCESSES

By GEORGE A. JAMES

(September 30, 1911)

Where water is used in other technical processes the importance of its analysis and proper chemical treatment is recognized, but it seems almost neglected in cyanide practice, where it stands in most vital relation. All natural waters contain impurities that have an influence on the results obtained by their use, and which can be corrected or improved by well known chemical and physical methods. There are various impurities found in natural waters, which have an effect on cyanide solutions, or ores; sulphates, chlorides, and nitrogen compounds of the various alkalis and metals, the carbonates and free sulphuric, carbonic, and hydrosulphuric acids. In addition to these, there is always organic matter in suspension and combined.

In cyanide practice many cases have occurred where preliminary tests have not agreed with later results. It is more than probable that such discrepancies can be traced to the water used. This indicates that the water should receive as thorough attention as the material to be treated, before a safe report on the results to be expected in practice can be given. A comparison of results obtained with distilled water, and the water to be used should always be made, and where differences are found, the reason should be sought and methods worked out for their correction. In experiments with waters, cyanide losses of two pounds have been found, and extraction losses of several per cent. My experience is that recovery in the majority of cases warrants the preparation of water before its use in any part of the cyanide process. The reasons for this can only be outlined, as the chemical reactions are obscure, and only a small part of the work necessary completed.

The first influence of water impurities is in preliminary washing of ore charges. The object of this is to remove soluble portions such as ferrous sulphate, copper sulphate, and free acid contents, which are destructive to cyanide solutions and retard percolation. The effect of ordinary water is to precipitate part of these impurities, very often adding other injurious compounds. Basic salts are formed (hydrates and carbonates), and as the sand acts as a filter medium, what organic matter there may be in the water is collected, to later destroy the leaching solution. Water containing acids may decompose iron compounds into a condition where they consume cyanide and form insoluble double salts. Sulphides are decomposed, resulting in sulphuretted hydrogen, another destructive agent for cyanide. This indicates the bearing water may have in this part of the process. In preparing stock solutions, annoying physical conditions and material loss of cyanide can be prevented by the proper treatment of water before the introduction of cyanide. Although my experiments have not reached the point where abso-

lute conclusions can be drawn, they indicate that cyanide destruction by carbonic acid is far greater than generally supposed. Natural waters contain 0.5 to 122 grains of this per gallon, and excessive cyanide destruction may result from this acid alone. Water containing free sulphuric and hydrosulphuric acid are obviously unfitted for cyanide solution until properly corrected, and the same applies in a lesser degree where water holds an appreciable amount of suspended organic matter.

The physical condition is also important in considering the independent treatment of water before its use in cyanide solutions. Where water contains a large excess of calcium hydrate as protective alkali, calcium carbonate is precipitated wherever it comes in contact with carbonic acid in the air or other solutions. This precipitate clogs filter-presses and zinc-boxes. Where water receives previous treatment, a large part of this trouble can be prevented. In the final washing of charges of ore, the chemical contents of water may produce the re-precipitation of metals and a further loss of cyanide. In conclusion, it may be stated that almost all waters can be cheaply corrected, and even slight improvements would warrant this correction.

All air contains impurities destructive to cyanide. Cyanide plants often exist in neighborhoods where, in addition to the carbonic acid always present, admixtures of sulphuric, sulphurous, and hydrosulphurous acid can be found in large quantities; in fact, the roasting process may be part of such plants. The need of removing such impurities before using air for agitation or aeration can be seen. There are simple methods of doing this.

CYANIDING BASE SILVER ORES

(October 7, 1911)

The Editor:

Sir—In reading over articles on fine grinding as they appear in the various mining journals, as well as enthusiastic metallurgical reports by firms supplying mining machinery, one may in time become convinced that all-sliming must, in some mysterious way, prove a solution of the treatment problem, even for the most basic and refractory ores. One feature, however, that is not always given the prominence which its importance demands, is that the finer the pulp is ground the more readily will it give up its undesirable base minerals as well as the precious metals.

A very base partly oxidized silver ore with which I recently had to deal, in Mexico, gave rise to some interesting results when I came to examine the working solutions after a month's run. The ore contained traces of copper and manganese, some zinc and antimony, considerable amounts of arsenic as mispickel, calcium and magnesium as carbonates, and alumina and silica; also large quantities of iron, both as sulphide and oxide, with occasional small amounts of ferrous iron as ferrous sulphate.

In assaying the solutions by the addition of lead acetate and throwing down the silver in a lead sponge by means of zinc shav-

ings, I found it necessary to filter and scorify before cupelling. A heavy white precipitate which formed on introducing a solution of lead acetate precipitated a portion of the silver from solution, and the white precipitate could not be satisfactorily collected with the lead sponge. 4 A.T. of working solution containing 18.3 mg. of silver to which were added 10 c.c. of a 10% lead acetate solution, yielded 3 gm. of white precipitate containing 0.3 mg. of silver. Hence, 1.60% of the silver was thrown out of solution by the white precipitate under the above conditions. I later found this white precipitate to be soluable in ammonium acetate, which would indicate a lead salt, so that it was possible to shorten the work somewhat by collecting the sponge and cupelling direct, though not advisable. Other results obtained were: total sulphur, 0.11%; sulphocyanides and nitrites, 0.18%; soluble sulphides, trace; copper, 0.09%. The remedy usually recommended for copper is the use of a zinc-lead couple in the zinc-boxes.

Tests with mercuric chloride gave unusual results. The addition of sufficient $HgCl_2$ to destroy all free cyanide in 4 A.T. of the working solution, containing 18.3 mg. of silver, yielded a precipitate weighing 4.5 gm. and containing 18.2 mg. of silver, thus indicating that the precipitation of the silver had been practically complete. In a second test a precipitate weighing 0.5 gm. contained silver amounting to 2.03 mg., which would indicate that the silver thrown out of solution was approximately proportional to the amount of the yellow precipitate found. Examination of this yellow precipitate showed that it was composed mainly of silky white mercurous chloride, occluded and masked by a yellow precipitate containing copper. The HgCl was reduced from the $HgCl_2$, due to the strong reducing action of the working solution. As HgCl is soluble in a KCN solution. why it should be formed and remain in suspension in this particular case is another interesting problem. Remedies which suggest themselves for this condition, are to remove the copper and to try to decrease the reducing power of the solution.

A test for reducing power of the working solution as compared with that of a newly made-up solution was made by oxidizing each with a standard N/10 $KMnO_4$ solution. Ten cubic centimetres of the new solution (KCN 0.235%) required 0.1 c.c. of the standard $KMnO_4$ solution, while 10 c.c. of the working solution (free KCN 0.35%) required 11.6 c.c. of the standard $KMnO_4$ solution, which means a reducing power of 116 times that of the new solution. Inasmuch as it is now thoroughly established that gold and silver require oxygen for solution, and this high reducing power can hardly be taken otherwise than indicating an absence of oxygen and oxidizing agents, it would seem to be no discredit to the cyanide process that an extraction of 70 to 85% could be maintained under such conditions, even though a higher extraction had been hoped for from the results of the preliminary examination.

J. E. CLARK.

Los Angeles, September 21.

ESTIMATION OF CALCIUM OXIDE IN LIME USED IN CYANIDATION

By LUTHER W. BAHNEY

(October 14, 1911)

*Lime is the alkali that is almost universally added to the solutions in the cyanide process of gold and silver extraction for maintaining the so-called protective alkalinity. It is produced by the burning of limestone. The value of lime for this purpose depends upon the percentage of calcium oxide contained, which is determined by three factors: (1) purity of the limestone used; (2) degree of the burning temperature and the period of burning; and (3) length of time of storage of burned material and the condition stored, and whether it has been damp or wet during the storage. These three factors render uncertain the quality of lime bought in the open market. In the United States, lime bought from reliable manufacturers, who thoroughly burn a pure limestone and deliver at once to the consumer from the kilns, may be of a fairly high and uniform composition; but in Mexico and Central America, where it is purchased from many small producers, who often start with a poor grade of limestone and burn it in small crude kilns with as little fuel as possible, the quality of the product is quite variable. It is therefore apparent that there is a great need for a rapid technical method for the valuation of the lime to be used in a cyanide plant.

The determination of calcium by the gravimetric method, with the necessity of determining also the proportion of CO_2, SiO_2, and Fe, requires too much time, and is usually out of the question for an isolated plant unprovided with a skilled chemist and the necessary apparatus. The calculation of all the Ca so found to CaO, although sometimes done, is manifestly inaccurate. Several methods of titration by means of a standard acid have been described, and no doubt give results sufficiently accurate for a technical method, but the objections to these methods are that they involve the preparation of a standard solution of some acid, usually decinormal hydrochloric acid, which cannot be weighed out, but must be standardized with some other standard solution. Solutions of the following acids have been used by different operators for standardization: sulphuric, nitric, hyprochloric, and oxalic. Oxalic acid is perhaps the most favorable for this purpose, because a standard solution can be prepared by weighing the solid acid and dissolving in water. The use of the solution employed to determine the alkalinity of the cyanide solutions has also been suggested. While the method of standardization with oxalic acid is open to the objection that the hydration of the acid may vary somewhat, yet it yields a solution sufficiently accurate for technical work, but so far as I am aware its use has not been suggested.

*Abstract of a paper presented at the San Francisco Meeting of the American Institute of Mining Engineers.

For the purpose of determining the feasibility of using oxalic acid, a solution of oxalic acid was made by dissolving 14.6068 gm. in enough distilled water to make a litre, this strength being recently suggested for determining the protective alkalinity of cyanide solutions. A decinormal solution of pure hydrochloric acid with distilled water was also made, and both were standardized with a solution of chemically-pure sodium carbonate. Pure CaO was prepared by grinding pure white crystals of calcite in an agate mortar and igniting the fine material in a platinum crucible over a strong blast until constant weight was attained. This oxide, cooled in a desiccator, was ground in an agate mortar to pass 200-mesh, and the percentage of CaO determined gravimetrically; the result was 99.98, as compared with the theoretical 100%. The CaO so prepared was used as a standard throughout the succeeding tests. Similar weighed portions were titrated with decinormal hydrochloric acid and oxalic acid, using phenolphthalein as an indicator, requiring 44.2 c.c. of hydrochloric acid or 44.6 c.c. oxalic acid to complete the reaction.

The weight of lime to be taken was calculated so that each cubic centimetre of oxalic acid solution would represent 1% of CaO, as given in the formula:

$$\underset{56.09}{\text{Lime.}} : \underset{x}{\text{Lime.}} :: \underset{126.048}{\text{Oxalic.}} : \underset{1.46068}{\text{Oxalic.}}, \text{ in which } x=650.$$

This weight, 650 mg., was used in all the tests, and the table shows the results, which are sufficiently satisfactory for a technical method. The titrations were made in the cold by introducing 650 mg. of the sample into a 300-c.c. Erlenmeyer flask containing 50 c.c. of distilled water, using phenolphthalein as an indicator.

RESULTS OF TITRATION TESTS FOR CALCIUM OXIDE, USING OXALIC ACID

Calcium carbonate present. %	Calcium oxide present. %	Cacium oxide determined. %
95	5	5.2
90	10	10.3
85	15	15.3
80	15	20.3
75	25	25.0
70	30	30.2
65	35	35.0
60	40	40.0
55	45	45.0
50	50	49.8
45	55	54.5
40	60	59.9
35	65	64.8
30	70	69.6
25	75	74.5
20	80	80.2
15	85	84.8
10	90	90.0
5	95	94.7
0	100	100.0

The results given in the table indicate that calcium oxide in the presence of calcium carbonate can be determined by this method with a fair degree of accuracy. Silica, present in most limes, does not interfere. Magnesia, also present in most limes in greater or lesser amount, is slightly soluble in water, and shows a faint reaction with the indicator; but it is of no value as an alkali in cyanide work and should not be shown in a determination of the available alkali in lime to be used for that purpose. Fortunately, the point where the alkalinity due to calcium oxide stops is readily recognized after a little practice, for the color is a vivid pink, while that of magnesium oxide is faint. Moreover, the color in the titration of magnesium oxide disappears with the addition of only 0.1 or 0.2 c.c. of oxalic acid solution, and returns very slowly and feebly, while that of lime is rapid and sharp. This is illustrated by the fact that a titration of pure calcium oxide requires only 5 minutes, while the same amount of magnesium oxide requires 3.5 hours. In order to test the oxalic-acid titration in the presence of magnesia, two samples of limestones containing magnesia were ground to 200-mesh, ignited in a platinum crucible to constant weight and titrated. The results obtained indicate that the magnesia does not interfere. Its presence can be judged by the behavior of the titration, and the approximate amount can be quite accurately estimated by continuing the titration, if one has the time needed.

Iron oxide in considerable amount is sometimes present in impure limes, and it obscures or masks the color of the indicator, but if the precipitate be allowed to subside the titration may be carried out to within 1% of the correct result. The determination of the amount of carbonate present in an imperfectly burned lime may be carried on as follows: Grind the sample to pass 200-mesh, weigh out 650 mg. and make the titration in the usual manner; call this result, available CaO. Ignite 650 mg. of the finely ground sample in a muffle or over a blast lamp, and make a second determination. Subtract the second result from the first, divide by 1.78, and the result will be the amount of carbonate present.

ELECTROLYTIC OXYGEN IN CYANIDE SOLUTIONS

By T. H. Aldrich, Jr.

(October 14, 1911)

*There are two conditions generally prevailing upon the earth—those within atmospheric influence, tending toward oxidation, and those away from atmospheric influence, tending toward reduction. Practically all mineral substances from mines of any depth are in a reducing condition. Since the cyanide process, in order to dissolve silver or gold, requires that the prevailing conditions under which it operates shall be oxidizing, and the material usually acted

*Abstract of a paper presented at the San Francisco Meeting of the American Institute of Mining Engineers.

upon being of a reducing character, it becomes necessary to supply oxygen to the solution carrying the cyanide. This oxygen is usually supplied through the medium of dissolved air in the solution, or by the use of one of various chemical compounds, which upon combining with the solution or the ore give off a part of their oxygen. In a dissolving race between the oxygen of the air and the reducing agents from the ore, if the reducing agents predominate, cyanide will not dissolve the gold from the ore. In many cases it will dissolve some of the gold, because in a mass of irregular shape some of the gold particles might be exposed upon the outside surface of a particle of rock; but if the solution had to penetrate through cracks, the side-walls of which were lined with reducing-agent-producing material, before the solution carrying oxygen could reach the gold it would have lost its oxidizing power. For this reason in many cases cyanide solutions will produce only an incomplete extraction of the gold or silver present.

It occurred to me that since water is composed of hydrogen and oxygen, if it be decomposed by the electric current, the hydrogen would bubble away and the oxygen would be carried by the solution. This was tried in December 1908 upon an ore carrying amorphous iron sulphides from which all the gold could not be dissolved by cyanide with simple air-agitation, no matter what the cyanide strength or how great the time. The process was tried first in an inverted bottle with the bottom cut out, the air being forced in through a glass tube in the cork to agitate the pulp. Two lead plates were inserted in the agitated pulp at the top. These plates were about 4 in. long and 0.5 in. wide, and $^1/_{16}$ in. thick. Through them was passed the current of an incandescent lamp, which gave about 0.25 ampere of current. The results were excellent. The value of the ore was $4 per ton. It was ground in a tube-mill so that 60% passed a 200-mesh screen. The value of the tailing, after 48 hr. agitation with air alone, was $1.25; but after agitation for 2.5 hr. with air and electrodes as described, the tailing was reduced to 40c. This typical result was verified many times, with uniformly good results.

In testing the solutions, a 2-lb. solution of cyanide is test 10. The alkali is tested on the basis of ten points over and above the alkali due to the cyanide, test 10 being a 2-lb. solution of caustic soda. The reducing agents were tested with a 1% solution of potassium permanganate, 1 c.c. of which in 10 c.c. of the solution, after acidulating, equals test 10, it being much easier to keep track of these solutions by simple numbers than by keeping the records in pounds per ton.

Numerous tests were made in order to determine a proper electrode. Lead was found to be the best material. Many other substances, such as carbon, worked very well, but with the alternating current, there being no consumption of the lead electrode, lead proved most satisfactory.

The following tests upon the working solution show the effect with iron and lead electrodes. All the tests were made at the same

time and with the same solution, using a direct current of 0.25 ampere.

Lead electrode:	KCN.	Alkalinity.	Dbl.	Reduction.
Before	8	+1	5	6
After 4 min.	12	+2	0	4
After 10 min.	12	+3	0	3
Iron electrodes:				
Before	8	+1	5	6
After 6 min.	7½	+6½	0	3
After 12 min.	6	+7½	0	2

(Showing destruction of the cyanide.)

There seems to be a regeneration of cyanide, and the process is certainly cheaper than any added oxidizer or even air-agitation of the solution. I found by numerous experiments that an alternating current was as effective as a direct, and had the additional advantage of giving no deposit on the electrodes at a lower current-density, and with lead there was no consumption of the electrodes even where the ore-pulp flowed over the electrodes.

As finally used in practice in January, 1909, a battery, supplied with alternating current, was placed in the barren sump. This battery consisted of 18 plates in series, each plate 6 by 6 in., with 110 volts between the two. The plates consumed 15 amperes, and produced sufficient oxidizing effect, or whatever other effect it may be, to keep the solution in condition to treat daily 40 tons of this ore. These plates, made of ⅛-in. sheet lead, were built so as to form hollow rectangles in sections, the rectangle being 6 in. high, 6 in. long, and 1.25 in. wide inside. The two ends were lapped at the top and holes punched. The plate was bolted to a paraffine plank 1 by 6 in. on the top side; 18 of these plates were connected in series. The distance between any two plates was ⅛-in., and, of course, the current would travel principally across the ⅛-in. gap, instead of around the 1⅜-in. gap from plate to plate. Lead wires were used from the surface of the solution down to the plates. The ore was ground in the tube-mills so that 60% would pass a 200-mesh sieve. Previous to using the batteries in the sump, the extraction in the tube-mill was 20% during grinding; after the batteries were used, the extraction in the tube-mill was 75%. The effect of the batteries seemed to build up in the solution gradually and to lose from the solution gradually when the operation of the batteries was discontinued.

During two months in the fall of 1910 the mill was working coarsely-ground, partly-oxidized ore carrying considerable sulphides. The water at the hydro-electric plant was low, and the use of the batteries was discontinued because the mill was driven by steam, and no arrangement had been made to supply alternating current from any but the hydro-electric plant. During this time the tailing on $4 ore went up to $1.25 per ton, and immediately after the rains gave sufficient water to drive the hydro-electric plant, the tailing assays diminished until 20c. per ton was reached on identically the same ore with the same head-assay; moreover, the reducing agents dropped from 16 to 4. The time occupied in getting the

working solution up to this condition was two weeks. I consider that this process owes its value almost entirely to the presence of oxygen due to electrolysis, putting the solution ahead in the race for the reducing agents and causing the gold and silver to dissolve in spite of the reducing agents. However, it does not stop the reducing agents from dissolving also, and although it produces solution of the gold in spite of the reducing agents, it does not help precipitation, and if the reducing agents are not decomposed by the batteries—and all of them are not—they build up in the solution rapidly to a point where zinc-shavings will not precipitate the gold.

Of course, in practice the cyanide solution contains reducing agents of many kinds. The electrolytic action seems to reduce the influence of some, but not all of them. For example, I experimented on some highly-graphitic ore, and whether the normally-poor extraction was due entirely to the graphite or not, I do not know; but the solution, after electrolyzing, gave a very much better extraction than before electrolyzing. The action seems to decompose the sulphocyanides and the soluble sulphides, but not the alkaline sulphides and all of the many others always present. A test on the electrolyzed solution 18 months after the batteries were installed showed:

	KCN.	Alkalinity.	Dbl.	Reduction.
Before	8	1	0	15
After 10 min. electrolysis......	8	1	0	15

showing that the solution remained practically the same, or was electrolyzed as much as was necessary. However, testing some of this same solution further by placing a piece of gold leaf on its surface and allowing it to float, the gold leaf was dissolved in 71 min. on the working solution and in 50 min. on the re-electrolyzed solution, showing that the additional electrolysis, although it had no apparent effect on the solution, gave an increased dissolving rate. Grease in the ore or on the surface of the barren sump seemed to dissolve very rapidly in the treated solution and slowly in the untreated solution. The tanks were tarred on the inside and coated with black oil on the outside, and more or less grease was frequently floating upon the surface of the solution where this effect was noticed.

Since the installation of this process it has treated successfully at this plant 25,000 tons of ore of all kind, oxidized, partly oxidized, and sulphides. Previous to the use of the batteries, in treating sulphide ores, the average cyanide consumption was 1 lb. per ton, in some months running as high as 1.1 lb. After the use of the batteries the average was 0.45 lb., running for some months as low as 0.23 lb. per ton of ore treated. I tried using batteries in the agitated pulp and in the solution, and found the result to be just as good if the plates were inserted in the barren sump as if inserted in the agitated pulp. The original lead plates placed in the barren sump are still there and in operation. They cost about $4 to insert originally, and were inspected after 26 months of practically continuous

service, and are today just as good as when they were first put in use. I have applied for no patents on this process and do not expect to, and anyone is free to use it. It should be a cyanide-saver, an accelerator, and a general solution-purifier.

TITRATION OF POTASSIUM CYANIDE IN THE PRESENCE OF POTASSIUM FERROCYANIDE

By W. D. Treadwell

(November 18, 1911)

Liebig's method for the direct titration of KCN with $AgNO_3$ is not applicable in the presence of $K_4Fe(CN)_6$. If, however, 0.1 gm. KI is first added to the solution, which should be about one-tenth normal and slightly alkaline, accurate results are obtained. The addition of NH_4OH, as recommended by Denigès and Sharwood, is not necessary. The disturbing influence of $Na_2S_2O_3$ may be overcome by increasing the amount of KI to 1 gm. I found a satisfactory explanation of the above facts in a study of the silver ion in the various solutions. This was detected from measurements of the electromotive force of a concentration cell consisting of one-tenth normal $AgNO_3$ and the solution to be tested. In the titration of pure KCN solutions with $AgNO_2$, it was found that upon the addition of the drop before the appearance of a turbidity, there was a sudden decrease in the electromotive force from 0.724 to 0.523 v., due to an increase in silver ions. Similarly the influence of excess NH_3, of $Na_2S_2O_3$, and $K_4Fe(CN)_6$ was found to be due to the formation of complex ions, and the lowering of silver ions below the concentration for precipitation of AgCN. Addition of KI caused in each case a turbidity due to the lesser solubility of AgI.—*Zeitschrift für anorganische Chemie.*

IRON AS A CYANICIDE

(March 16, 1912)

The Editor:

Sir—There is nothing new in the statement that small fragments of iron in the ore will consume cyanide during lixiviation. However, a word regarding such a common cyanicide may not be amiss. Julian and Smart make the statement that iron dissolves in KCN, yielding potassium ferrocyanide; one part of iron consuming 7 parts of KCN, the reaction also consuming oxygen, which is required for the dissolution of the precious metals. In a recent cyanide test this cyanide occurred in such a way that it made its presence known. The table concentrate from an ore carrying a small amount of sulphide, after standing a few hours, showed rusty streaks which gave the concentrate the appearance of an oxide rather than a sulphide.

One gram of this rusty material was withdrawn by means of a magnet and treated 16 hours in a cyanide solution, with a resulting consumption of 34 lb. KCN per ton of material. Investigation before cyanidation showed that only about 10% of this rusty material was metallic iron; the rest was simply discolored gangue clinging to the iron. The rust gave this pulp the appearance of containing limonite and, as nearly as could be determined by intensity of discoloration, none of the rust was dissolved. A consumption of 34 lb. KCN per ton of solids treated is 1.7% of 1000 mg. or 17 mg. KCN.

Since the entire consumption is chargeable to metallic iron, which amounted to 100 mg., and iron consumes cyanide at the rate of 1 to 7, it can be seen that no appreciable amount of iron was dissolved, though the cyanide consumption due to its presence sounds large. Since this magnetic material constituted only about 1% of the concentrate, it is evident that the cyanide consumption due to iron in the concentrate would have been 0.34 lb. Going one step farther, it is seen that the consumption in the ore would have been negligible in a laboratory test.

One thing worthy of note is that, as shown by the calculations, after iron consumed cyanide at the rate of 34 lb. per ton it had only begun, and that in long-period agitation (6-day, for instance) it is important to select crushing machinery that will contribute a minimum amount of metallics.

WILL H. COGHILL.

Evanston, Illinois, March 1.

IRON IN MILL PULP

By A. McA. Johnston

(October 12, 1912)

*At one time ordinary cyanide practice in the mills of the Rand included concentrating to a rich product which was cyanided separately and which yielded, after many days' treatment, a fair proportion of its gold to the extractor boxes and incidentally added a considerable quantity of ferro and sulpho-cyanide to the solutions. The practice has of late years been superceded by treatment of this product with the sand and the presence of iron in cyanide solutions has thus not been so noticeable. Incidentally, however, with the introduction of coarse screening and the beneficial effects of the tube-mill circuit (Dowling, 'Rand Metallurgical Practice,' pp. 123-129), a state has arisen, which, although present previously, has been emphasized by the existence of this circuit. A quantity of iron is present in the circuit, and remains there, it may be, for a very long period.

Iron is obtained in the crushed product from several sources. These may be tabulated as follows:

*Excerpt from a paper submitted to the Chem. Met. & Min. Soc. of S. A., on August 17, 1912.

Ore.

(*a*) Metallic iron in the ore itself.

In a deep deep-level ore, iron has been found in the metallic state amounting to 0.013% of the ore.

Mine.

(*b*) Iron from wearing of the drills—either fine or in chip form.

(*c*) Iron from broken drills.

(*d*) Iron from bolts, nails, or pieces of scrap introduced underground. Hammer-heads are also a not unknown product in the ore sent to the mill.

Mill.

(*e*) Iron from wear of steel bins, chutes, and feeders.

(*f*) Iron from wear of mortar-box liners.

(*g*) Iron from wear of shoes and dies.

This has been stated to be about half a pound of iron per ton of ore crushed (Dowling, 'Rand Metallurgical Practice,' p. 132).

(*h*) Iron from screens used in mill.

(*i*) Iron from keys, etc., accidentally dropped in mortar-box.

Tube-mill.

(*j*) Iron from wear of feeders and of end liners.

(*k*) Iron from iron or steel liners, rails, or pegs used in the tube-mill.

(*l*) Iron from nails and scrap accidently dropped into the circuit.

The iron from these various sources may be classed into two kinds, namely, that in a very fine state of division, due to its formation by abrasion, and that in coarser or pellet-like form, due to chipping, breakage or accidental inclusion in the pulp. In mills where fine crushing is still in vogue, the percentage, per ton of pulp, of iron in a fine state will, naturally, exceed that found in mills where coarse crushing has been adopted, while again the removal of the larger pieces of iron from the mortar-boxes, along with the die sand, owing to their inability to escape therefrom in the pulp, will in the former case lower the amount of this class of metallic iron in the tube-mill circuit.

The specific gravity of the iron, when compared with that of the ore, renders it more liable to remain in the tube-mill circuit. It may continue there for some time, till at last, through some fortuitous circumstance, the finer portion is carried to the cyanide plant with the pulp, while the coarser particles continue their weary round till abrasion has rendered them in a fit state to follow. This iron in the tube-mill circuit affects the working in various ways. These may be sumarized as follows:

(*a*) The coarser particles are liable to scour the plates and cause loss of mercury. The mercury may later on be recovered

from the launders, pumps, or traps, or it may be lost by flouring and consequent escape into the cyanide plant.

(*b*) In conjunction with the pyrite, the finest particles are likely to lie on the plates and so cover the amalgam that amalgamation will be interfered with and more frequent dressing of the plates necessitated.

(*c*) Again, owing to the heavy nature of these particles, it may be advisable to supplement the flow of water over the plates or increase the grade of fall in the plates so as to diminish the tendency to interfere with amalgamation.

(*d*) The metallic iron is very apt to rob the plates of mercury by cohesion. This can be easily verified by visual inspection of the larger pieces, while microscopic examination of even the finest particles shows occasional specks of mercury.

(*e*) Owing to its retention in the tube-mill circuit, the iron causes additional expense in tube-mill work, due to its weight and the necessary addition of water to increase the flow.

(*f*) As a considerable quantity of this iron is in pellet form, its inclusion in the tube-mill product means less efficiency in the crushing process, for the pebbles, instead of crushing the sand, expend energy in flattening the iron. An examination of the product will show justification for this claim.

The fine iron which has overflowed into the cyanide pulp will, when the cyanide solution is pumped on the charge, be in one or other of the three states: metallic iron, ferrous oxide, or ferric oxide; the two latter being partly present in combination with water as hydrates. Iron in rusting forms an external covering of ferric oxide, with, underneath, a layer of ferrous oxide in contact with the metallic iron. Oxidation continues until the metallic iron has been converted into ferrous oxide and this eventually into ferric oxide. It is well known that ferric hydrate or ferric oxide does not form ferrocyanide with cyanide, and generally speaking, this covering over the iron protects it from the action of cyanide. The rubbing of the particles of sand on this, however, may break the protective coat so that the cyanide eventually reaches part, at least, of the ferrous oxide and metallic iron. Though the action of cyanide on iron vats or extractor boxes is so small as to be negligible from a practical standpoint, we must remember that the iron entering the plant with the pulp is in a fine state of division and therefore the more liable, not only to react with the cyanide, but to be changed into ferrous hydrate or ferrous sulphate, which are easily converted into ferrocyanide.

Thus the fine iron entering the cyanide plant may form, in contact with the cyanide solution, sodium ferrocyanide as follows:

$$6NaCN + Fe + 2H_2O + O = Na_4Fe(CN)_6 + 2NaOH + H_2O$$

$$6NaCN + FeO_2H_2 = Na_4Fe(CN)_6 + 2NaOH$$

$$\text{and } 6NaCN + FeSO_4 = Na_4Fe(CN)_6 + Na_2SO_4.$$

The sodium ferrocyanide thus formed acts prejudicially on the solution of the gold, since it is a reducer and depletes the solution of oxygen (Mac Farren, 'Cyanide Practice,' p. 43).

In addition to this effect of fine iron and its derivatives on cyanide, it is to be remembered that ferrous sulphide is produced when grinding pyrite between iron surfaces. $FeS_2+Fe=2FeS$, which latter is detrimental to gold-bearing cyanide solutions (Caldecott, 'Rand Metallurgical Practice,' p. 383). The same reaction would naturally take place in the tube-mill, when the sand and the metallic iron are being crushed by the pebbles.

These influences may appear of little importance in ordinary work and may be overlooked, but if it is remembered that in a solution entering the strong extractor boxes there has been used 326 lb. of sodium cyanide to hold in combination 0.02% $Na_4Fe(CN)_6$ in 1000 tons, the advantage of removing as much iron as possible from contact with cyanide solution, when it is in such a state that it will be attacked, is evident.

To eliminate this iron from the tube-mill circuit a magnetic separator was installed under the direction of A. C. Holtby, assistant mechanical engineer to the Consolidated Gold Fields. After running a few days a representative sample, or as near such as could be obtained, owing to the nature of the material, was taken, and the following tests conducted thereon. It was found that a considerable quantity of non-magnetic material had been caught, owing probably to mechanical adherence in the wet state to the magnetic portions.

Microscopic Test.—A microscopic examination showed the finer portion of the sample to contain its gold in minute nuggety form, mostly adhering to comparatively large pieces of quartz. A considerable amount of pyrite was present in the finest part, though this portion contained little, if any, amalgam or mercury.

Assay Value of 14.50 Dwt. Per Ton

Grading Test Aperture	Percent.	Cumulative, %	Assay Value, dwt.
+ 0.276	0.47	0.47	trace
+ 0.122	7.10	7.57	27.4
+ 0.103	2.87	10.44	20.0
+ 0.063	5.77	16.21	18.6
+ 0.033	12.90	29.11	37.2
+ 0.020	15.40	44.51	14.4
+ 0.015	11.17	55.68	12.6
+ 0.010	26.03	81.71	20.0
+ 0.016	13.76	95.47	44.4
− 0.006	4.53	100.00	136.0
	100.00	Calculated	29.12

This grading test shows that the plus, 0.010-in. aperture amounting to 81.71% in weight, contains 41.1% of the gold, and the minus 0.006-in. aperture material, 4.53% by weight, contains 21.1% of the gold.

Magnetic Test—A portion of the sample was separated by means of a magnet into magnetic and non-magnetic parts.

	Per cent.	Assay Value, dwt.	Total Gold in Sample, %.
Magnetic	73.5	1.6	8.3
Non-Magnetic	26.5	48.8	91.7
	100.0	Calculated 14.1	100.0

It will be seen from this test that were it possible entirely to separate the magnetic from the non-magnetic, 91.7% of the gold would be retained in the latter.

Cyaniding Tests, both before and after roasting, gave very low extraction—from 2 to 4% of the gold. In these tests a very considerable amount of sodium ferrocyanide was formed. An analysis of another sample somewhat similar to the above showed:

	Per cent.
Iron (metallic)	87.4
Ferrous oxide	3.7
Ferric oxide	1.2
Pyrite	2.5
Silica	4.8
Moisture (at 100° C.)	.02
	99.8

After the magnetic separator had been working about three months, a large sample, amounting to about a ton in weight, was taken for experimental purposes. This was piled in a heap on a flat clean surface where it was exposed to the action of the atmosphere. The weather was dry and cold during the three months of the test. The pile was turned over daily, while each alternate day it was thoroughly wetted, and once a week the water added contained 10 lb. of commercial salt. During the earlier part of the treatment considerable heat was evolved, due to the oxidation of the iron and pyrite. Some temperature readings about 6 to 10 in. below the surface indicated from 160 to 170°F. At the end of four weeks a sample was taken from the lot and analyzed after being quickly dried on a hot plate.

A. Grading.	Per cent.	
+0.006 in.	1.07	(chiefly metallic iron)
−0.006 in.	98.93	
	100.00	

Analysis.	Per cent.
Silica	9.14
Alumina	trace
Lime	trace
Magnesia	trace
Iron (metallic)	1.59
Ferrous oxide	9.96
Ferric oxide	73.24
Pyrite	3.06
Moisture (at 100° C.)	1.29
	98.28

From this it will be seen that the bulk of the iron present as metallic iron has been converted into iron oxide. A portion of the sample was amalgamated by being shaken in a bottle with mercury for 12 hours.

	Dwt. per ton.
Assay value of original	6.1
Assay value of residue	3.8
Extraction by amalgamation	2.3
	= 37.7%

The treatment of continued exposure to the atmosphere and with occasional wetting with water plus one spraying with a solution of 0.01% sulphuric acid, was continued for four more weeks. The analysis and amalgamation tests on this are given in B. After a further four-weeks treatment with spraying and turning over, the pile was again sampled and analyzed. The results are given in C.

Grading.	B, %.	C, %.
+ 0.006 in.	2.70	0.94
− 0.006 in.	97.30	99.06
	100.00	100.00
Analysis.		
Silica	11.13	10.51
Alumina	trace	trace
Lime	trace	trace
Magnesia	trace	trace
Iron (metallic)	3.04	1.04
Ferrous oxide	8.06	8.11
Ferric oxide	71.82	75.23
Pyrite	3.00	3.30
Moisture (at 100° C.)	1.60	1.00
	98.65	99.19

The larger percentage of plus 0.006-in. pieces of metallic iron in B is due to the inclusion of one or two large particles. It is to be noted that all the sulphur has been calculated as pyrite. Portions would of course be present as sulphate and also as ferrous sulphide. The main object of the analysis was to obtain a general idea of the degree of oxidation of the iron.

C. A portion of this sample—that is, after 12-weeks exposure —was taken and amalgamated in a bottle for 12 hours with occasional shaking.

	Dwt. per ton.
Assay value of original	6.1
Assay value of residue	3.9
Extraction by amalgamation	= 36.0%

The residue from amalgamation was then cyanided (without previous drying) in a large bottle with occasional agitation. Time of treatment 36 hours, plus 24 hours for water washing.

	Dwt. per ton.
Assay value of washed residue	1.7
Extraction by cyaniding	2.2
Total extraction by amalgamation and cyaniding	4.4
	= 72.1%

The cyanide solution, 3 parts of solution to 1 of ore by weight showed:

Originally,	0.165% KCN, 0.065% NaOH.
After 36 hours treatment,	0.093% KCN, 0.014% $K_4Fe(CN)_6$, 0.009% KCNS.

The general conclusion to be drawn is that it is advantageous from a working, amalgamating, and cyaniding point of view to remove the metallic iron from the tailing. It can easily be oxidized by wetting and exposure to the atmosphere, but care should be taken that it be turned over and disintegrated daily, or at least each alternate day, so as to avoid the formation of lumps, until the greater part of it has been converted into the oxide. Generally speaking, this means until there is practically no more heat generated. Without this, the mass will form into hard cakes or pieces difficult to break up and slow to oxidize. Even with daily handling, little nodules form, the interior of which may consist of metallic iron and ferrous oxide, and may remain thus for some considerable time. The presence of silica, of course, helps materially in rendering the mass more porus and hence the more easily oxidized. When large pieces of iron in the form of bolts, nuts, or nails are present, the process would necessarily require a more lengthened period than is shown above.

It will be noticed that salt was used to assist in the oxidation process. It is a moot point whether this is necessary with this class of material. Personally, I should expect almost as quick results without its use. It is to be noted in conclusion that during the initial stages of the running of this separator, nearly a ton of magnetic iron, with silica and pyrite mechanically removed, was taken from the circuit daily. This has gradually decreased until after over six months running the daily yield of magnetic iron from a mill crushing 2500 tons per day is about 400 to 600 lb., or one pound per 5 tons of ore.

Summary.—The main points to be noted may be summarized thus:

(*a*) Iron is chiefly obtained in the tailing from the crushing plant, and is mostly present as small pieces, smaller than a pea, or as fine as powder.

(*b*) Iron in this stage interferes with amalgamation and with the re-crushing in the tube-mills, and also consumes cyanide in the leaching vats.

(*c*) When removed from the pulp, this product can be easily oxidized by exposure to the atmosphere, wetting, and turning over.

(*d*) Failing any better means of disposal, this oxidized product will, if fed into the circuit, yield an average percentage of its gold to amalgamation and cyanidation.

REFRACTORY MANGANESE-SILVER ORES

By Will H. Coghill

(June 1, 1912)

In the month of January, 1910, an ore sample was brought to my laboratory for treatment to determine the amount of silver amenable to cyanidation. I noted that it contained a small amount of MnO_2, but gave no thought to this feature and put the ore through the ordinary routine tests, finding that the cyanide process could not be successfully applied on a commercial basis, and a report was made stating the refractory nature of the ore. It then occurred to me that something had been written about refractory manganese-silver ores in Mexico, and that this sample might belong to that class, so at my request the sample was left for further investigation. The client also submitted a report made by a metallurgist in Mexico, who stated that this ore belonged to a class known as refractory manganese-silver ores; that there were a great many orebodies of that nature in Mexico; that the silver seemed to be associated with MnO_2 in a way that had never been identified; that wet concentration processes failed to segregate the silver with the heavy grains; and that contrary to the rule, these ores were refractory at the surface and became more amenable to treatment upon depth. The metallurgist further stated that he knew of several mines which were idle because there was no process known by which the silver could be extracted economically and that it was yet a mystery why some ores contained manganese dioxide and were amenable to cyanidation, the dioxide even aiding by acting as an oxidizer, while others contained dioxide, which seemed to be the cause of the refractoriness. Here, indeed, is an eccentric ore, for, though the silver seems to be in the dioxide, a wet concentration of the dioxide fails to reveal a corresponding concentration of the silver. It is also a rule generally supposed to be without exception that gold and silver orebodies are more amenable at the surface than at depth, but here the conditions are reversed. Again, MnO_2 does not aid cyanidation, as is popularly supposed.[1]

With this information at hand, rather elaborate experiments were made to determine the properties and possibly identify a new mineral, as well as others of a commercial nature. Samples from three localities were investigated. The first is that designated as the Arizona ore, and two samples sent from Mexico at my request are designated as the Durango ore and the Jalisco ore.

[1]A private communication from J. B. Empson in reply to an inquiry made nearly two years after this investigation was begun, reads in part: "As you rightly conclude in your letter, there are several properties in Mexico that are unable to treat these ores profitably by cyanide owing to the amount of manganese present. So far as I am aware at the present moment, nobody has been able to shed any light on this matter."

TABLE I.

SIZING TEST—ARIZONA ORE

Mesh.	Grams.	Total sample, per cent.	Ag per ton, ounces.	Total Ag, per cent.
On 80........	15.2	25.7	13.80	18.6
80-150........	13.2	22.3	15.60	18.4
150-200........	6.8	11.5	17.15	10.4
Through 200........	23.9	40.5	24.20	52.6
	59.1	100.0		100.0

(1) A sizing test on the Arizona ore (see Table I) resulted in a concentration of silver with the smallest grains, 52.6% passing 200 mesh with 40.5% of the ore, while only 18.6% remained on 80 mesh with 25.7% of the ore. The method of crushing cannot be stated, as the ore was received in a powdered condition. (2) In another sizing test, 56% of the ore remained on 200 mesh and after cyanidation assayed 12.40 oz. silver per ton (calculated from the assay of the classified products shown in Table II), and was treated in Thoulet solution (KI and HgI_2) to separate the mineral grains according to their specific gravity. Three were secured, namely, grains with a specific gravity greater than 3.10, with specific gravity greater than 2.69 and less than 3.10, and grains with specific gravity less than 2.69. (See Table II.)

TABLE II.

SPECIFIC GRAVITY TEST—ARIZONA CYANIDE TAILING

Specific gravity.	Grams.	Total sample, per cent.	Ag per ton, ounces.	Total Ag, per cent.
+ 3.10........	10.30	26.7	29.00	63.2
2.69-3.10........	9.90	25.6	7.50	15.7
—2.69........	18.40	47.7	5.40	21.1
	38.60	100.0		100.0

It is quite apparent that the refractory silver is not enclosed in quartz grains, because inspection of the table reveals that the products consisting principally of quartz assay only 5.40 oz. per ton, while the heavy grains are high in silver. Table I shows a concentration of the silver with the smallest grains, and Table II shows a concentration with the heaviest grains. This proves conclusively that at least a great percentage of the refractory silver is in a soft heavy mineral. (3) Persistent grinding and agitating in cyanide extracted only 34% of the silver. (4) Only 32% of the silver yielded to hot strong HNO_3 upon a ten minute treatment of the original sample. (5) A similar treatment with dilute HCl with subsequent ammonia treatment extracted 69%. (6) A like treatment with strong HCl recovered 85% of the silver. Thus far it is indicated by (6) and Tables I and II that the refractory silver is chemically combined with or incorporated in an oxide which is heavy and soft. Now hydrochloric acid is the common solvent for oxidized, and

nitric the common solvent for unoxidized minerals. I cannot quote this from an authority, but it is only natural that if nature had made a mineral insoluble by oxidizing it, it can be put into solution with a reducing agent—HCl. If it becomes insoluble through reducing conditions, it should be treated with an oxidizing agent—HNO_3—to put it into solution. It is not known how many oxides were dissolved by the HCl, though it was indicated by the color reaction that MnO_2 was decomposed. So it seems that the silver was exposed to the decomposition of an oxide. Suspicion rested upon the MnO_2. An attack was made upon it with a dilute solution of H_2-SO_4 and salt. (7) It dissolved quickly, and tailing from a previous cyanide test yielded 82% of the refractory silver. The action of the sulphuric acid and salt is two-fold: it dissolves the manganese dioxide and in so doing liberates chlorine which chloridizes the silver. The consumption of sulphuric acid proved to be excessive, because it attacked the calcite in the ore. Treatment under pressure in a stoppered bottle prevented the sulphuric acid from attacking the calcite, but the pressure also prevented the evolution of chlorine, which was an essential part of the reaction; therefore the dioxide remained undissolved. (8) Now, H_2SO_3 is a good solvent for chlorine and was added with the H_2SO_4 and salt in the next test in which the charge was agitated under pressure. The reaction was complete in a few seconds, the dioxide color being replaced by the color of the iron oxide. (9) Further tests proved that H_2SO_3 attacks MnO_2 with such avidity that the other reagents had no opportunity to participate. E. M. Hamilton has stated[2] that he used H_2SO_3 as a reducing agent, but I was hunting for a solvent and did not have the results of his investigation in mind at that time. Though very few textbooks mention the fact, it has been known for a long while that MnO^2 is soluble in H_2SO_3 and these reagents have been used to make manganous dithionate (MnS_2O_6), some $MnSO_4$ being formed. There is probably no other case in which a mineral is as soluble in a cold dilute acid; a 1% solution being sufficient to dissolve powdered MnO^2 in a few seconds. (10) It was surprising to find that after dissolving the dioxide in H_2SO_3 such a large percentage of silver was amenable to amalgamation. A sample which passed 200 mesh and assayed 24.20 oz. gave 63% extraction by amalgamation when treated in the jar mill for twelve hours. (11) A sample of cyanide tailing yielded 45% of the otherwise refractory silver during a two-hour treatment. (12) Also cyanidation of the original sample gave an 85% extraction, but this method is out of the question on account of the soluble salts formed by the acid treatment. The conduct of the dithionate toward the potassium cyanide was not investigated, but the $MnSO_4$ consumes cyanide and must be entirely neutralized by an alkali. This cannot be accomplished economically on account of the great amount of solution required. As the sample was used up, the investigation was reluctantly discontinued. It seemed quite possible to work out an

[2]*Mining and Scientific Press,* December 4, 1909.

amalgamation process in which amalgamation would proceed under pressure in a closed barrel with fine grinding and acid treatment. A pressure of about 10 lb. per square inch would be necessary to prevent the H_2SO_3 from spending its strength on the calcite, a little salt would quicken the mercury, and copper sulphate in the presence of salt would aid in dissolving the silver. As the ore contained only about 2% manganese, the consumption of sulphur on the basis of manganous dithionate being formed would be 47 lb. per ton of ore. (13) Attention was then directed to a sample from Durango, Mexico, which had been classed with the refractory manganese-silver ores. . It contained 10% MnO_2 and assayed 21.85 oz. silver and 0.16 oz. gold per ton. It was stated by the superintendent of the mine from which this sample came that the ore yielded from 40 to 60% of the silver by cyanidation and that the amount of refractory silver varied directly as the amount of MnO_2 in the ore. (14) A sizing test gave a decided concentration of the silver with the smaller grains. (15) The silver displayed the same refractory disposition toward HNO_3 as did the Arizona sample, and (16) a large percentage was dissolved by HCl treatment. (17) Tests by amalgamation, hyposulphite lixiviation, and cyanidation proved the utter futility of attempting to extract the silver without fine grinding. Even MnO_2, though it dissolves in a few seconds in dilute H_2SO_3 when crushed to pass 200 mesh, may resist the acid for an hour when coarser than 40 mesh. (18) After crushing in an Abbé pebble mill, a siding test gave 22% on 200 mesh, 78% through 200 mesh. It was upon this material that the following tests were made. (19) The amount of gold dissolved was not influenced by the presence or absence of MnO_2.

Table III represents a good deal of work and does not indicate a commercial method of treating the ore, but when studied from a scientific standpoint becomes interesting. The tests, which were not made in the order shown, cover a period of several months' investigation, which was carried along with other work. It was a great satisfaction to see that these fit together with so much harmony. For instance, numbers (23) to (26), inclusive, show a steadily increasing percentage of extraction, depending on the time, and that more than 18 hours' treatment would probably be useless.[8] Note that the mercury failed to combine with the silver in (21) and collected more than one-half of the silver in (20). In (24) the superiority of potassium and ammonium hyposulphite over the soda salt should be noted. I am not aware that anything has been written on this, but tried them just to satisfy my curiosity, with no idea of using them commercially. Calcium thiosulphate was also tried, but results were low. Sodium thiosulphate is understood to be the hypo salt referred to in these tests unless otherwise specified.

[8]It was revealed later that all the results in this column are low on account of the ore being a little too coarse for all MnO_2 to dissolve when given the acid treatment.

TABLE III.

LIXIVIATION TESTS—DURANGO ORE

Reagents.	No.	Per cent extracted: Without acid treatment..	Per cent extracted: After acid treatment.	Duration of test in hours......	Remarks.
Mercury	20		51.0	18	
Mercury	21			8	
Sodium thiosulphate	22		25.0	16	
	23		56.0	1	
Thiosulphate and mercury......................	24		81.0	12	Ammonium thiosulphate, 78.0%.
	25		81.0	12	Ammonium thiosulphate, 78.0%.
	26		81.5	18	
	27	21.0		6	
	28		78.0	9	
Thiosulphate and copper sulphate...............	29		81.5	48	Solution changed once.
	30	43.0		16	
	31	36.0		20	Salt to make 5% solution.
	32		66.0	7	+200 mesh sizing test.
	33		83.0	7	−200 mesh sizing test.
Thiosulphate, copper sulphate, and mercury.....	34		69.0	7	Salt to make 5% solution.
	35		73.5	7	
	36		81.0	9	
	37		87.0	12	
	38		71.0	8	
Copper sulphate, salt, and mercury..............	39	21.3		8	
Ammonia and ammonium chloride.............	40	0.0		8	
Acetic acid	41	13.5		8	2% solution.
Ammonium persulphate	42	20.5		8	2% solution.
Ammonium persulphate and mercury............	43		84.0	12	Agitation in cyanide.
Potassium cyanide	44	44.3		18	Solution changed once.

When the strong reducing power of a thiosulphate solution is taken into account and the strong oxidizing power of MnO_2, it would seem to be useless to expect the thiosulphate to last long enough to dissolve any silver, but tests show that neither thiosulphate nor sodium sulphate are decomposed by the MnO_2. In the tests shown in Table III, 1 assay-ton of the sample was treated in a jar-mill. Thiosulphate, when used, was about 1½% of the solution and copper sulphate about $^1/_5$%. The jar-mill was simply pint glass fruit jars with rubber stoppers and lids screwed on to hold the stoppers in place. For abrasive materials marbles and small flint pebbles were used. The jars were fastened with axes parallel to a rotating shaft when grinding in solution was required and were fastened with axes at right angles to the shaft when merely agitation was desired, in which case they would be turned end-over-end. All the jar-mill tests were made by grinding in solution except number (43). The solution used in (29) did good work. It is the Russell 'extra solution' of cuprous-thiosulphate, and it is claimed by the inventor that it exerts a greater solvent action on the metal, the sulphide and the sulpharsenide and sulphantimonide, that is, all but the chloride, than does the ordinary thiosulphate solution. The late O. Hofmann, in his excellent work on hydrometallurgy of silver, speaks quite disparagingly of the Russell process, stating that it is superior to ordinary thiosulphating only in laboratory tests with very strong (32%) solutions. My solutions were fairly dilute (1½%), and the superiority of the extra solution is again noticed in (36) and (37) when compared with (24), (25), and (26). (31) and (34) contained salt in addition to the other reagents. Comparing them with (30) and (25), respectively, shows that salt was a detriment when used with the extra solution. This closely parallels the condition that the extra solution must contend with in practice, as the ores are generally given a chloridizing roast in which there is an excess of salt. However, much of the salt is leached out by the base-metal leaching which precedes the thiosulphating. It may be that salt simply reduces the activity of the reagents, as does sodium sulphate in ordinary thiosulphating, but it is more likely that salt reacts with the copper sulphate (these reagents were added in the solid state and at the same time), making cupric chloride and sodium sulphate, thus preventing the copper from combining with the hypo to make the double salt and introducing sodium sulphate, which is objectionable.

In using mercury in these tests it was not the aim to develop a commercial process in which mercury would be used with these salts, but it was used as a collector to simplify laboratory work. Mercury collects silver completely from the thiosulphate, and can be assayed for silver by the crucible method, giving accurate results when a flux of low fusion temperature is used. If the fluxes are not of the right sort losses will occur. The use of the glass separating funnel[4] makes the separation of the mercury from the tailing a simple matter.

[4]Will H. Coghill, *Mining and Scientific Press*, July 9, 1910.

It is a question whether (32) and (33) show the importance of excessively fine grinding or that there was more refractory silver in the harder and larger grains. The reader should bear in mind that grinding was continued throughout the lixiviation tests, and it would seem that the grains would all be crushed to an impalpable powder whether or not they were 200-mesh or —200 at the beginning. Subsequent experiments indicated that some of the MNO_2 remained undissolved in the 200-mesh material; this no doubt accounts for the inferior work in (32). The high extraction in (37) as compared with (28) and (29) is probably due to the united forces of the mercury and the solution, where the mercury at once takes up the larger metallics while the solution is dissolving the smaller and more refractory particles. (38) is an imitation of the patio process in which cupric chloride is formed, which is said to be an aid to amalgamation. (41) is just a chance shot taken on account of Thomas Crowe's statement,[5] that "ammonium persulphate is a good solvent for silver," and shows that it is a solvent in this case at any rate.

The amount of sulphur required probably prohibits a preliminary treatment in sulphurous acid. Assuming the formation of manganous dithionate, the theoretical amount required is 151 lb. per ton of ore. (45) In a test on 5 gm., sulphur was consumed at the rate of 150 lb. The absence of calcite is a favorable feature which the Arizona ore did not possess.

Table IV shows results of a series of calcination and chloridizing roasting tests on the same ore. One assay-ton of the ore was taken, and when a chloride was used it was thoroughly mixed with the ore, making 9% of the charge. The temperature is uncertain. Accurate records of the temperature of the muffle were made with a Bristol thermo-electric pyrometer, but it cannot be stated how closely they correspond to the temperature in the scorifier. The jar-mill was used in the lixiviation as in Table III.

Very little has been written on the chloridizing roasting of non-sulphide ores, and it would therefore make a good field for some energetic investigators. In all ordinary chloridizing roasting the ores contain sulphides, usually of iron and copper, which become oxidized to sulphates. These sulphates react with the salt making sodium sulphate, which is a stable compound under existing conditions, base metal oxides and nascent chloride, which combines with the silver. There being nothing in this ore from which to make a sulphate, $MnSO_4$ was added (the idea being that in commercial work a small part of this ore might be treated in H_2SO_3, making a sulphate, then mixed with the untreated portion and roasted) in (50) and (54), but as (53) and (54) were treated side by side, there is pretty positive evidence that the sulphate was a detriment. In (52) there is positive proof that silver is chloridized by chloridizing roasting this non-sulphide ore, the temperature at which this reaction begins being about 617°. In attempting to ac-

[5]Researches on Cripple Creek Tellurides. *Mining and Scientific Press*, Sept. 15, 1909.

TABLE IV

CALCINATION AND LIXIVIATION TESTS, DURANGO ORE

	Calcination			Lixiviation							
	Reagents.	Temperature, 0°C....	Duration of Test, hr.	Reagents.	Duration of Test, hr.	Oz. Ag per Ton: Recovered........	Oz. Ag per Ton: In Tailing........	Oz. Ag per Ton: Lost..............	Recovery, %........	Extraction, %.......	Remarks.
46	None	900?	¾	Extra	8	2.12			9.7		Sintered.
47		850	¾	solution	8		17.70			19.0	Not sintered.
48	Salt	900?	¾	and	6	8.56	3.74	9.55	39.2	83.9	
49	Flour	900?	1¼	mercury	6					58.4	KCN tailing from (44).
50	Salt and manganese	625	2	Thiosulphate	10	15.16	5.60	1.09	69.5	74.4	
			2¼	and mercury							
51	Sulphate	617	2¼	Ammonia	1	5.12	16.00	0.73	23.4	26.8	
52	Salt	617	2¼		1	16.40	5.36	-0.09	75.0	75.5	
53		617	2¼		4	17.30	4.40	0.15	79.3	80.0	
54	Salt and manganese sulphate	617	2¼		4	16.60	5.30	none	76.0	76.0	
55		583	1		10	9.74	10.32	1.79	44.6	52.8	
56	Salt	583	3		10	9.60	8.94	3.31	44.0	59.1	
57		583	5	Thiosulphate	10	9.60	9.64	2.61	44.0	60.6	
58	Calcium chloride....	620	3	and mercury	10	13.14	7.56	1.15	60.2	65.4	
59	Barium chloride....	620	3		10	3.88	17.54	0.43	17.8	19.7	
60	Magnesium chloride	620	3		10	3.80	15.82	2.23	17.4	27.5	

count for the chlorination of the silver it may be assumed that there is a reaction between the salt and the nascent oxygen from the MnO_2, resulting in the formation of free chlorine and sodium oxide, the former chloridizing the silver and the latter combining with the silica to make slag. That the ore was not sintered in several tests in which chlorination took place does not disprove this theory, because it is probable that a large portion of the salt was dissipated in the form of a vapor, leaving only a small portion to undergo such a decomposition. Vapor could be seen rising from the charge during some of the tests, but it was not determined whether it was free chlorine or sodium chloride. A. V. Bleininger, in a private communication, is inclined to think that "silver would become chloridized at about 650°C., at which temperature clay dedydrates and is capable of reacting with many chemical substances." In (55), (56), and (57) titration failed to reveal a loss of chlorine, and a low extraction followed, though some of the silver was probably chloridized. Test (58) indicates that calcium chloride is a chloridizing agent. This is extremely interesting, because it is generally supposed that calcium oxide will steal chlorine from silver during chloridizing roasting. Here it should be mentioned in connection with the Arizona ore that the calcite would be a factor to be reckoned with in case chloridizing roasting were to be tried. The great loss of silver as shown by (48) is to be expected when chloridizing at a high temperature. In (49) 1 assay-ton of the cyanide tailing from (44) was mixed with the 40 gm. of flour and calcined in a covered crucible at about 900° for 1¼ hours, then a lixiviation test was made by grinding in extra solution plus mercury, resulting in an extraction of 58.4% of the refractory silver. There is no doubt that the MnO_2 was here reduced to Mn_3O_4, or a lower stage of oxidation, and that a longer period of calcination or lixiviation, or both, would have resulted in a much higher percentage of extraction. Comparing this with (46) and (47), in which there was calcination without flour, shows that the flour is a very active reagent during calcination, either by exerting a reducing effect or preventing sintering. I do not know of any published data on the dissociation temperature of the native oxides of manganese, but wish to note here that the dissociation temperatures of the artificial oxides have been determined,[6] namely,

MnO_2 at 530°C. breaks down in air to Mn_2O_3
Mn_2O_3 at 940°C. breaks down in air to Mn_3O_4

I have some data on the dissociation temperature of the native MnO_2 which I will publish later.

The large amount of silver in the tailing in (51) is due to the ammonia being partly neutralized by the manganous sulphate.

[6] *Z für Anorg. Chem.*, vol. 57, 1908.

The mercury (½ assay-ton) in (59) was badly sickened, and in (60) was not as active as it should have been.

Let us now consider an ore from Jalisco, Mexico, which contained 18.70 oz. silver, 0.05 oz. gold, 10% manganese (15.8% MnO_2), and which in the preliminary tests conducted itself in such a way that it seemed justifiable to put it in the class with the Arizona and Durango samples. It is unlike the Arizona ore in that it did not contain calcite, but like the others in that it contained manganese as an MnO_2, quartz, limonite, and refractory silver. Now these ores were all so stained and discolored that it was nearly impossible to distinguish between a piece of quartz and MnO_2, but it was noted that if a sample was screened and that part remaining on 10 mesh given a wash in water, followed by a brief wash in very dilute H_2SO_3 all stains would be removed and hand sorting become a simple matter. Table V shows results of such a test on a grab sample.

TABLE V

CLASSIFICATION BY HAND SORTING, JALISCO ORE

	No.	Gm.	% of Total	Oz. Ag. per ton	% of Total Ag.
Mainly dioxide........	61	4.90	19.3	25.06	36.6
Mainly quartz	62	20.50	80.7	10.40	63.4
		25.40	100.0		100.0

Here it is noted that a product assaying 25.06 oz. silver is secured from a sample assaying 13.25 oz. silver. This certainly makes it desirable to repeat the test on a larger scale. (See Table VI.)

The sample of the Jalisco ore tested, amounted to about one-third of the sample screened and could not be assayed because sampling would have involved crushing, but calculating from the table (IV) gave 16.99 oz. silver per ton and 8.16% manganese. Three products were made; the first looked like pure pyrolusite, the second contained small pieces of dioxide and quartz cemented together, which, if the reader will excuse a Joplin term, was 'chatty'; the third looked like nearly pure quartz. (The term quartz is used for simplicity and includes many light-colored silicates.) For want of better names, I will call these 'concentrate,'

*Extensive experiments are being made for the purpose of securing a higher extraction from the manganese ores of El Favor mines, Jalisco, Mexico, and several promising methods are now being tried, the best of which seems to be from the work of the Massachusetts Institute of Technology where tests indicate that a much higher percentage can be saved through magnetic concentration. This process is now being thoroughly tried by Walter Neal, according to the June, 1913, report of El Favor Mining Co.—(Editor.)

TABLE VI

CLASSIFICATION BY HAND SORTING, JALISCO ORE

	Number	Gm. of each	Per cent of total	Oz. Ag per ton	Per cent of total Ag	Ag, %	Mn, %	MnO_2 (calculated), %	Fe, %	Limonite (Calculated), %	Insoluble in aqua regia, %	Total % determined.
Concentrate	63	112.5	11.8	34.00	23.6	0.12	40.6	64.1	1.5	2.5	33.2	99.9
Middling	64	78.0	8.2	33.50	16.2	0.12	18.8	29.7	2.4	4.0	65.0	98.8
Tailing	65	760.0	80.0	12.80	60.2	0.04	2.3	3.6	1.5	2.5	90.9	97.0
		950.5	100.0		100.0							

'middling,' and 'tailing,' respectively. The analyses of these products bring to light some astonishing relations. The amount of silver found in the concentrate and tailing was about as expected, but the discovery of a middling containing practically the same amount of silver as the concentrate, and only about one-half as much dioxide, would make one feel that he has got some light on a problem which has perplexed so many. In fact, this explains at once why J. B. Empson reported of one of these ores that in a concentration test the tailing contained as much silver as the concentrate, and why E. M. Hamilton, after attempting to concentrate a refractory manganese-silver ore, stated that "the tailing assayed rather higher than the concentrate." Now since the tailing, as shown by Table VI, contains only 12.80 oz. silver and 71.1% (3.6+2.5+6.5) of the material in the middling is the same as that in the tailing, it is apparent that there must be some very rich grains in the middling. Thus the part that I had expected to throw away became the most interesting, and indicated that the amount of refractory silver varies, not as the amount of dioxide, but as the amount of cementing material and is therefore at the contact of the dioxide and quartz. Sixty hours' grinding in cyanide, during which time the solution was changed twice, extracted 61% of the silver in the tailing (65). Under similar treatment the concentrate (63) yielded on 13.4% and the tailing from this test was given subsequent treatment as shown in Table VII.

TABLE VII

JALISCO ORE

0.5 A. T. Concentration (60-hour treatment in KCN):

			No.	Mg. of Ag.	% Total Ag.
Solution			66	2.28	13.4
Residue (treated in H_2SO_3)	Solution		67	0.10	0.6
	Residue (15 minutes treatment in dilute NH_4OH)	Solution	68	0.68	4.0
		Residue	69	12.44	73.3
		Error		1.50	8.7
				17.00	100.0

Here it seems that the silver is slightly soluble in sulphurous acid or ammonia, but subsequent tests proved that the amount is small and depends on the conditions. The 'error' may be due to the difficulty experienced in fluxing the residue containing the manganese salt, to the silver being put into such a state that some of it adhered to the jar, or more likely to a loss incurred by splitting the sample into four parts.

The most interesting feature of this table at the time the tests were made was the insolubility of the silver in ammonia. This

was supposed to prove that cement silver was not present, but later it was found upon making a test on silver leaf that the metal is practically insoluble in ammonia, though some texts say that it is soluble. During a two-days' test on silver leaf in ammonia with frequent aeration, the silver finally decomposed, forming a dark-brown cloud and later a dark-brown flaky precipitate, but this change was too slow to be of consequence. There seemed to be reason to suspect the formation of cement silver (resulting from partial precipitation) in the sample as it was indicated that the acid had exerted at least some solvent effect on the silver. Now, if it had all been converted into a sulphate or sulphite and remained as such, it would have been soluble in the amount of solution used, but since the greater part of it remained with the residue, it seemed quite probable that it had been dissolved and then partly precipitated by the iron of the iron oxide, thus making cement of colloidal silver. All fears of cement silver being formed by the action of iron oxide upon the silver salt were dispelled by the test (70) following: A sample of the ore was treated with sulphurous acid to which was added a known amount of a solution of silver nitrate (the nitrate was of course converted at once into a sulphite or possibly a sulphate). The test was allowed to stand 15 minutes after the dioxide had all dissolved, thus exposing the silver salt to the reducing action of the iron, and was then filtered and washed. The assay of the residue showed that none of the silver was put into an insoluble state. (71) The insolubility of elemental silver in sulphurous acid was indicated by a test on silver leaf, the silver leaf remaining undissolved and without discoloration for two days in a stoppered bottle in the presence of that acid. It then took on a color similar to that of bornite and remained in this condition for

TABLE VIII

JALISCO ORE

	No.	Mg. Ag recovered	% Total Ag	Gm. of product assayed	Oz. Ag per ton of product assayed
0.2 A. T. concentrate treated in H_2SO_3:					
Solution	72	....	....	....	....
Sand	73	0.52	7.6	1.498	10.10
Slime	74	6.00	88.3	0.460	380.00
Error		0.28	4.1		
		6.80	100.0		

several weeks. Therefore, it is seen that elemental silver is soluble in neither NH_4OH nor H_2SO_3; that a sulphate or sulphite is not formed, for if it were it would remain soluble and not be decomposed by the iron oxide, making cement silver.

Before investigating Table VIII, which contains valuable information, a certain property of native manganese dioxide must be considered. When the dioxide is dissolved in sulphurous acid a light colored nearly gelatinous precipitate is formed which remains in suspension with great persistence. This consists probably of barium and calcium sulphates formed by the reaction of sulphuric acid in the sulphurous acid upon the oxides which exist as impurities in the dioxide. Now, in dissolving the dioxide in these ores the same sort of a precipitate forms, except it is colored a brick red due probably to the limonite in the ore.[1] (The solution, on account of conditions which were not determined, may become red, due probably to ferric sulphite, and in turn be decolorized upon bottling.) This brick-red slimy material is called 'slime' in Table VIII. The heavier particles consisting mainly of silica are called 'sand.' These were separated very easily by decantation.

The amount of silver in the sand (73) corresponds closely to that in the tailing (65), but the concentration of silver in the slime (74) sufficient to give an assay of 380 oz. per ton gives a clue that is valuable, for the number of mills treating sand and slime separately runs up into the hundreds and in no case is there such a segregation of silver in either sand or slime. This indicates, then, that here we have to deal with silver in a form that has never been previously met in an ore.

After demonstrating that silver is released during the dissolution of the dioxide, forming such a light slimy product, it seemed best to make a series of tests to determine its chemical deportment, that is, to find whether it belongs to class *x, y, z* or *w*.

$$\text{Silver.}\left\{\begin{array}{l}\text{Artificial}\left\{\begin{array}{l}x\ \text{Elemental}\\ y\ \text{Combined}\end{array}\right.\\ \text{Native}\left\{\begin{array}{l}z\ \text{Elemental}\\ w\ \text{Combined}\end{array}\right.\end{array}\right.$$

If it is an artificial product, it would be expected to partake of the properties of any precipitate and form slime. If it is still in the native metallic state, remaining unchanged while dioxide dissolves, sliming is not to be expected. If in the native combined state, sliming is quite possible, for it is likely that the mineral stephanite (brittle silver) or any mineral as brittle as chalcopyrite, for instance, would segregate in the slime, though not to such an extent.

[1]In fact, admixtures of manganese oxides and limonite are known to occur in all proportions, ranging from manganiferous limonites to manganese ores carrying only a trace of iron.

Table IX

JALISCO ORE

			No.	Mg. Ag.	% Total Ag.
0.2 A. T. middling (treated in H_2SO_4 and washed):					
Solution			75	0.18	2.7
Re idue (treated 15 minutes in acetic acid, decant and wash)	Solution		76	0.12	1.8
	Residue (15 min. vigorous shaking with Hg)	Mercury	77	0.00	0.0
		Residue	78	5.75	86.0

In Table IX (76) shows the insolubility of the silver in acetic acid, and in so doing, it certainly eliminates Ag_2O from *y* and probably *w*, though I om not informed in regard to its action, if such a native compound exists. The rest of the table deserves notoriety such as a criminal receives, therefore I will ask the reader not to place much emphasis on (77), which shows that the silver was refractory toward mercury. However, it led me to make some calcining tests, which are shown in Table X.

Table X

CALCINATION TEST, DIOXIDE TAILING, JALISCO ORE

(Calcined ½ hr. after H_2SO_4 treatment, then amalgamated 2 hours.)

No.	Temp.	Per cent Extracted.
79	640	14.0
80	757	9.0
81	885	7.0

The so-called 'dioxide tailing' used in this table and tests following is the result of thoroughly cyaniding a hand-sorted sample containing about one-third dioxide. It assays 20.25 oz. silver. The amalgamating was done by grinding with mercury. An increased temperature of calcination is shown to have reduced the percentage of extraction. After failing to amalgamate by the methods outlined in tables IX and X, what could one do but turn to wet reagents? The results were as follows:

(82) Oxalic acid and oxalic plus sulphuric—common solvents for MnO_2—converted a part of the silver into an oxalate and sulphate, respectively. In several tests the dioxide was first dissolved in sulphurous acid and then given the above treatment. The results were all erratic and proved nothing except a slight solubility in all cases, the highest extraction amounting to 72.8 per cent.

(83) 'Hypo' was tried both in the presence of sulphurous acid and after the dioxide had been dissolved, but proved also a poor solvent, though as high as 50% of the silver was extracted in one case.

In a series of tests, in all of which the dioxide had been dissolved, a solution (84) of $KI+HgI_2$ with subsequent treatment in NaCl extracted 47% of the silver; (85) agitating 15 minutes in 20% HCl cold chloridized 46.3%; (87) a 10% HNO_3 solution (cold) extracted 66.7% in 45 minutes; (86) fourteen hours' agitation in 5% H_2SO_4 extracted 7.4%; while (88) a hot solution of the same strength extracted 75.4%; (89), a concentrated brine solution, failed absolutely.

I will not burden the reader with a further recital of tests which show the perversity of this mineral, though I have before me, tabulated, the results of 40 assays which were made by a student to determine the dissolving power (after the dioxide had been decomposed) of the following reagents which I will repeat in the order of decreasing dissolving efficiency: HNO_3 hot, HNO_3 cold, $KI+HgI_2$, oxalic$+H_2SO_4$, oxalic, 'hypo,' H_2SO_4, NaCl, NH_4OH. Suffice it to say that the best is a poor solvent. What, then, was my surprise to find, (90), that a 3% cyanide solution dissolved 89% in 15 minutes, and (91) 96.3% in 1 hour, a dissolving rate comparable with that of silver chloride in trough lixiviation. Of course, I was quite aware that cyanide was a good solvent, but such rapid action is not the ordinary rôle of KCN. Now, what about the criminal referred to in (77) which indicated that mercury was a solvent? This test was repeated and made (92) in the same manner as before, with the addition of an ammonia treatment which followed the acetic acid. The mercury extracted nearly all the silver. In short, it developed that a small part of one cubic centimetre of dilute acetic acid was sufficient to put the silver into such condition that an amalgamation was impossible. No visible films were formed on the mercury. Taking into account the relative amount of materials, it would seem that the silver, not the mercury, was to blame. Just to clear up the matter and show that mercury was a willing solvent, test numbers (93) and (94) were made during two and four hours, giving, respectively, 80.7% and 87.6% extraction. These amalgamation tests now harmonize perfectly with those made on the Arizona ore. The lesson, then, to be emphasized in connection with Table X is evident: do not try to amalgamate after calcining the acid-treated ore, though the temperature be kept low. There seems to be an incipient fusion of the silver or gangue, or both, which prevents amalgamation.

Ordinarily it would be possible to recapitulate after such a series of tests and to draw accurate conclusions, but in this case, if one conforms to the statements in books on chemistry and metallurgy, he must concoct some unheard of compound. However, after correcting an erroneous impression in regard to ammonia, everything except the peculiar conduct toward acetic acid and extreme solubility in cyanide points to *x*—elemental, artificial, or pre-

cipitated silver. Now, any millman would view with disgust the statement that precipitated silver is readily soluble in cyanide and cite good authority for his belief. Though I would not disagree with him in regard to silver precipitated from cyanide solutions, I believe that the rule is not of general application,[2] and submit Table XI. The amount of ore taken in each case was 0.2 A. T.

TABLE XI

DIOXIDE TAILING, JALISCO ORE

No.	Mg. Ag Added.	Kind of Solvent. (Preliminary treatment in H_2SO_3)	Hr. Agitated	% Extracted
95	..	Thiosulphate 3%	16	40.7
96	25	Thiosulphate 3%	16	45.0
97	..	Extra solution ('hvpo' 3%, $CuSO_4$ 1%)	16	97.8
98	25	Extra solution ('hypo' 3%, $CuSO_4$ 1%)	16	80.0
99	..	$CuSO_4$ 3%, NaCl 3%, then NH_4OH	8	76.8
100	25	$CuSO_4$ 3%, NaCl 3%, then NH_4OH	8	89.0
101	..	KCN 1.5%	1	96.3
102	25	KCN 1.5%	5	85.0
103	..	Amalgamation	2	80.7
104	25	Amalgamation	2	90.0
105	..	$Fe_2(SO_4)_3$ 3%, H_2SO_4 0.5% cold	10	11.1
106	25	$Fe_2(SO_4)_3$ 3%, H_2SO_4 0.5% cold	10	46.3

The '25 mg. Ag added' was precipitated silver in the form of a very fine powder, precipitated from an ammoniacal solution with ammonium formate. It was added to the sample and given several hours' grinding in a jar-mill previous to the preliminary treatment in sulphurous acid.

The table is intended to show the relative solubility of precipitated silver and the silver in the residue remaining after treating the ore in H_2SO_3. Incidentally, it emphasizes a point already made: that of the greater efficiency of the 'extra solution' when compared with thiosulphate—compare (95) and (97). The same thing is noted in (96) and (98) where the precipitated silver is dissolved; (99) and (100) show how readily the precipitated silver and the silver in the residue is chloridized by cupric chloride. The ready solubility of the silver in cyanide as shown by (101) has been referred to before and is put in this table so that it can be compared with (102) which shows that precipitated silver has about the same solubility as the silver in the residue. The amalgamation tests in (103) and (104) compare favorably. In (105) and (106) there is a lack of the harmony that is not shown in the preceding

[2]Louis Janin, *Eng. & Min. Jour.*, Dec. 29, 1888, showed that cement silver is quite soluble in dilute standing cyanide solutions, but less soluble in a concentrated solution. He does not say how the cement silver was prepared.

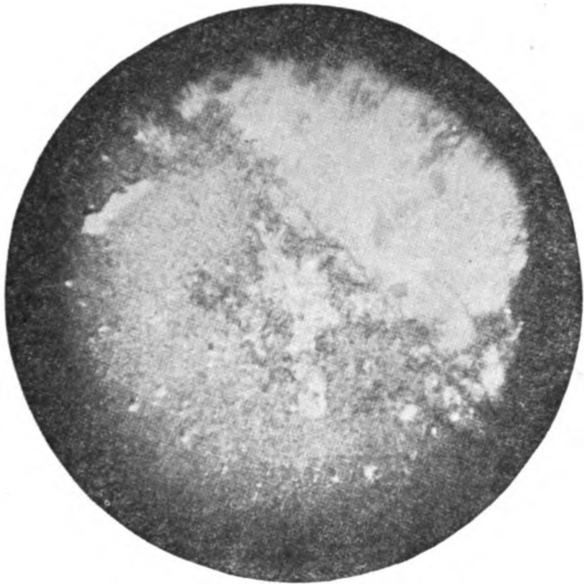


Fig. I. Contact of Quartz (to the right) with dioxide of Manganese (to the left).

tests. H. N. Stokes[3] has tested the solubility of silver in ferric sulphate and finds that the reactions involved are reversible, silver being dissolved and in turn precipitated upon change in temperature. Consequently, I am inclined to ignore the tests with ferric sulphate and to conclude that since the silver in the residue dissolves in the other reagents at so nearly the same rate as does the 'Ag added', the silver in the sample belongs to the group x and might well be called cement or precipitated silver.

In Table XII the subject is attacked from rather a different viewpoint. After considering the condition of the silver in the dioxide, it seemed desirable to learn in what kind of dioxide grains the silver occurs. The sample was screened and washed as explained in connection with Table V, and six products were made by hand sorting. The calculated assay value is 18.42 ounces.

The middling or 'chatty' grains (108) running high in silver corroborate the statements made in connection with (63) and (64) and the high silver in (110) and low silver in (111) show that the botryoidal and stalactitic pieces of dioxide give a higher assay in silver than does the very flaky and cleavable dioxide. Or, putting it another way, (108) and (110) show that the chatty and stalactitic grains are the richest. The iron oxide (112) assayed higher than was expected. It was not absolutely free from dioxide; this may account for it. In order to follow up this study of types of grains, a number of micro-sections were made; those of the types included

[3]*Economic Geology,* Vol. I, 1906, p. 649.

TABLE XII

JALISCO ORE

(Classification by hand sorting.)

	No.	Gm.	% of Total	Oz. Ag per ton	% of Total Ag.
Concentrate	107	15.00	7.5	21.70	8.8
Middling	108	44.50	22.1	39.00	46.7
Tailing	109	135.00	67.1	9.60	35.0
Dioxide (botryoidal and stalactitic)	110	1.85	0.9	134.00	6.6
Dioxide (flaky or cleavable)	111	1.60	0.8	18.20	0.8
Iron oxide (red and yellow	112	3.20	1.6	25.20	2.1
		201.15	100.0		100.0

in (108) and (110) receiving the most attention. The slides were all examined by an experienced petrographer and nothing of interest noted. Now the study of opaque minerals, such as the oxides of manganese, of course, involves the use of reflected, not transmitted, light. I felt certain that a 'chatty' piece should reveal something characteristic in the dioxide and near the contact with the quartz. I therefore tried reflected light, both natural and artificial with and without a vertical illuminator. After several hours of such experimenting, I succeeded in getting focused on some dark brown spots in the steel-gray dioxide and near the quartz contact. I say spots instead of mineral grains, because they did not show a definite contact, but blended into the dioxide. The spots were found along the contact and reminded one of the way drift-wood may be strewn along a lake shore. After these things were found, their identification became much easier, they being most easily seen in a moderately bright reflected sunlight. Wishing to secure photographs, I visited the metallographers for the Illinois Steel Co., who spent about two days with me polishing sections and using their metallographer's outfit for magnifying and photographing. This work was a failure. L. H. Weld, who has made a specialty of photographing micro-sections, then came to my rescue. After several hours adjusting and waiting for a light of the proper intensity, a negative (113) was secured. It has been intensified to bring out the dark-brown spots which photographed darker than the polished dioxide. Though the dioxide is actually the darker, it does not seem so in the picture, because of the

minute flakes which reflect the light and give it the appearance of micaceous hematite. I cannot say positively that the picture shows an unidentified silver mineral, but believe that it does. It seemed to me that a procedure similar to the following should give proof: Remove the cover glass, dissolve the balsam on the upper surface with alcohol, then submerge in sulphurous acid and let remain undisturbed until the MnO_2 dissolves. Take from the acid bath and inspect the residue under the microscope and place it in a cyanide bath. Silver would be readily dissolved and its disappearance when noted by means of the microscope would be positive proof of silver. This was tried, but was a failure, possibly on account of the residue of limonite which veiled the silver. However, I think that persistent efforts along this line would be successful. I studied one slide of the stalactitic dioxide (110), but found nothing similar to that noted above.

Resume

1. A screen sizing test, Table I, showed a decided segregation of the silver in the smaller, therefore softer, grains.
2. Treating the cyanide tailing, Table II, in a Thoulet's solution showed a segregation of the silver in the heaviest grains.
3. About 60% of the silver in these ores (3), (44), (64), and (65) could not be dissolved in KCN.
4. Nitric failed absolutely (4) to dissolve the refractory silver. A silver mineral that could not be dissolved by either KCN or HNO^3 was probably unheard of until noted recently by E. M. Hamilton.
5. The silver can be chloridized (5) and (6) by HCl, which dissolves MnO_2 and possibly other oxides, thus indicating that the silver is in an oxide.
6. The silver can be chloridized (7) by sulphuric and salt, a common solvent for MnO_2.
7. Sulphurous acid is a remarkably active solvent (8) and (9) for MnO_2, but does not dissolve the silver.
8. After dissolving the dioxide in H_2SO^3 (10) and (11) the silver can be amalgamated or cyanided, though great care would be necessary to prevent undue consumption of cyanide by the soluble manganese salts resulting from the acid treatment. If carbonates are in the ore the acid treatment must be under pressure.
9. The gold (19) is amenable to cyanidation.
10. Many of the common solvents for silver, Table III, are effective only after the dioxide is dissolved.
11. In (24) the order of increasing efficiency of the thiosulphates was, calcium, sodium, potassium, and ammonium.
12. The Russell 'extra solution' (28) might prove that it has virtue where thiosulphating is practiced, but the conduct of the NaCl (31) and (34) must be watched.
13. One pound of S as SO_2 (13) and (45) will dissolve about 1.3 lb. MnO_2.

4

14. Manganous sulphate (54) does not aid in chloridizing roasting as do iron and copper sulphates.

15. The chloridizing roasting (52) of ores containing MnO_2 is possible and is an absolutely new field as far as published data are concerned, even $CaCl_2$ (58) being operative. Attention has been called to the fact[4] that NaCl and KCl can be mixed in such proportions that they will melt at 135° below the melting point of pure salt. A chloridizing roasting test with this mixture might prove instructive.

16. Comparing (49) and (46) shows that flour has an effect during calcination worthy of investigation. A reduction in a Stetefeldt furnace fired by gas and a deficiency of oxygen might accomplish the same results and obviate the introduction of objectionable constituents.

17. Hand sorting, though ordinarily impossible, can be done with perfect ease, after a wash in dilute H_2So_3, thus removing the dioxide stain.

18. Table V shows that the silver is largely in the dioxide, and Tables VI and XII show positively that the amount of refractory silver does not vary as the amount of dioxide, as formerly supposed, but as the amount of dioxide in contact with quartz, that is, as the amount of chatty material. This condition makes wet concentration absolutely impossible.

19. I have shown in (70) and (71) that silver does not form a salt with the reagent when these ores are treated with H_2SO_3.

20. Table VIII shows what I consider an unheard of condition of silver in the ore, namely: Upon dissolving the dioxide in H_2SO_3 the silver is held in suspension in the solution with the limonite and other slime-forming compounds.

21. Acetic acid (77) makes this residual silver refractory toward mercury, but an additional treatment in ammonia (92) removes the obstacle, and it can be easily amalgamated.

22. Table X shows that the residual silver cannot be amalgamated after calcination at a low temperature. Calcining for ½ hour at 321° below the melting point of silver being sufficient to bring about this refractory condition.

23. Many reagents (82) to (89), inclusive, were poor solvents for the residual silver, and Table XI shows a striking similarity of cement silver to the silver in the residue resulting from treating the ore in H_2SO_3. The avidity with which KCN (101) attacks it indicates a unique mode of occurrence of the silver.

24. Now, it has been shown in Tables VIII and XI that the silver in the residue resulting from treating the ore in H_2SO_3 conducts itself unlike the silver in any ore familiar to myself, at least, and in a way nearly identical to the porous precipitated silver with which it was compared; that in (70) and (71) silver does not form a salt with the reagent (H_2SO_3) used in the preliminary treatment; that a micro-section (113) shows dark-brown splotches in the part of the dioxide in which it had been previously predicted (64); and

[4] *Eng. & Min. Jour.*, March 2, 1912, p. 439.

(108) that there was a segregation of silver. Therefore it seems evident that the cement silver is the result of silver mineral being decomposed contemporaneously with MnO_2, this silver mineral being the dark-brown splotches above mentioned. To give the composition of this mineral would be mere conjecture. Its association with such a highly oxidized compound would suggest the oxide; if an oxide, then why does it not dissolve in acetic or sulphurous acid?

25. The outlook for investors in mines of this type is not good. Cyaniding without preliminary treatment is absolutely impossible. In the way of preliminary treatments, the one by sulphurous acid must be considered, though it involves purchasing sulphur in amounts of about one pound for every pound of dioxide in the ore, then it would have to be burned, and the resulting fume could not be handled without enough escaping to raise a protest among the millmen. The presence of carbonates such as calcite would increase the consumption of sulphur. The expense of neutralizing after such a treatment preparatory to cyaniding would be considerable. The alternatives of amalgamation and thiosulphating remain. The relative merits of the processes could be determined only by further experiments. If there were a good demand for artificial MnO_2 the market could be supplied by applying electrolytic precipitation to the manganese solutions. The production of a cheap by-product, brimstone, at the smelters would lend favor to the H_2SO_3 treatment. Chloridizing roasting, I understand, has been practiced on ores of this type, but cost of fuel and silver losses militate against success. Chloridizing has been mentioned under 15.

26. I submit this problem to both geologist and metallurgist: to the geologist for him to identify this evasive silver mineral and account for the mode of occurrence; and to the metallurgist for him to map the bodies of refractory manganese silver ores so that investors may receive fair warning, and after giving such warning proceed with the investigation of methods of extraction.

ACTION OF MINERAL SULPHATES AND ARSENATES ON CYANIDE SOLUTIONS

ANDREW F. CROSSE

(July 6, 1912)

*It has long been known that both sulphate of magnesia and sulphate of lime decompose cyanide solutions. The mineral kieserite is not rare, and has the composition $MgSO_4+H_2O$, and according to 'Comey's Dictionary of Chemical Solubilities,' is but slowly dissolved in cold water. Hydrated sulphate of calcium or gypsum is also common, but it interests us more because sulphate of calcium is produced when the acidity in tailing and slime is neu-

*From the *Journal* of the Chem. Met. & Min. Soc. of S. A.

tralized with lime, and I have also found alumni taken up in cyanide solutions, so that probably some aluminum sulphate or double sulphate was in the ore. The first point I would like to mention is that an ordinary analysis of a mineral or sample of the ore to be treated by the cyanide process is nearly valueless.

For instance, take a common example—a certain ore contains iron, sulphur, magnesium, and other elements, according to the analysis. This analysis is no help to the cyanide manager. What he wants to know is, what will take place when the finely crushed ore comes in contact with an ordinary cyanide solution? The Mount Morgan ore near Barberton is a good example. I have seen analyses of this ore, but as far as I can remember, none of them showed any sulphate of magnesium; magnesia was mentioned and so was sulphur, and as this ore is highly pyritic, I did not think of the presence of sulphate of magnesia until I began treating the ore. I found on making a preliminary test that the ore, digested in warm water, gave a neutral reaction. In my first experiments I used an ordinary amount of caustic lime, and found the protective alkali gradually disappearing, and eventually free HCN was being given off. Then I digested some of the finely ground ore in warm water, and found magnesium sulphate in the solution. I determined the sulphuric and hydride in solution, but I found out in subsequent experiments that more SO_3 was taken up when using carbonate of soda solution, or cyanide solution. I consider the best method to determine the amount of soluble sulphates present in an ore, is to take 100 gm. of finely ground ore and shake it up with 500 c.c. of warm water containing say 2 gm. of carbonate of sodium of known percentage of Na_2CO_3. I prefer to leave the mixture all night and shake it up again next morning, and then titrate the Na_2CO_3 left, taking 50 c.c. of the filtered solution, and the amount of carbonate of sodium used per ton can be easily calculated. In the case of the Mount Morgan ore it was 10 lb. per ton of ore, but the same ore after treating with warm water only required 4.4 lb. of Na_2CO_3 per ton of ore to precipitate the $MgCO_3$ and $CaCO_3$ from theaqueous solution, so that even if the ore were washed, it would still decompose a cyanide solution; therefore carbonate of sodium is necessary. This is quite an important point for if the sulphates were easily removed by water, this would take place during crushing in the mill.

There is a peculiar vein found in the Machavie mine on the Black Reef, between Potchefstroom and Klerksdorp. This pyrite is in a globular form, and there is a large amount of graphite in this ore (the Mount Morgan ore also contains garphite). Some of the tailing from that mine after washing four times decomposed cyanide of potassium (without using protective alkali) at the rate of 0.64 lb. per ton. The unwashed sand, however, under the same treatment, decomposed 3.2 lb. of cyanide of potassium per ton of ore. This ore contained sulphate of magnesium.

I made a long series of experiments with various proportions of CaO in solution (lime water) added to a mixture of equal parts

of decinormal pure cyanide of potassium solution, and decinormal sulphate of magnesia. This seemed very nice and simple, and I thought that I should get some definite results. There was a time not so very long ago when it was imagined that all chemical reactions could be nicely expressed in equasions, but unfortunately the more the mysteries of the physical world are studied the more it is found that nature does not arrange things to suit preconceived ideas. A nice equation is the following:

$$2KCN + MgSO_4 + 2H_2O = 2HCN + Mg(OH)_2 + K_2SO_4.$$

When equal portions of the decinormal solutions of the above salts are mixed, a precipitate of magnesium hydrate is formed, and free hydrocyanic acid is given off, but I doubt whether the reaction is as complete as the equation indicates. I thought that if I added the lime-water, gradually increasing the amount till no more hydrocyanic acid was liberated (when tested by a drop of nitrate of silver solution placed on the lower side of a glass cover on a beaker) I should arrive at some definite result, but as sulphate of calcium was formed, which also dissociates cyanide of sodium or potassium, it made things complicated. Again, as more CaO solution was added, weakening the cyanide of potassium solution, hydrolysis took place. I found, however, that all the magnesium became precipitated as a hydrate. Then I also found that free hydrocyanic acid was given off, notwithstanding that, using the ordinary test, protective alkalinity as CaO was present. I am not satisfied that when a cyanide solution containing sulphates of lime and magnesia is tested by a nitrate of silver solution, that a correct result is obtained. The solutions I used were pure; but of course in practice the solutions contain zinc, which complicates matters still further. The ordinary practical man will shrug his shoulders at all this, but after all, the question is how to get over the decomposing action of sulphate of magnesium in the cheapest way. Carbonate of sodium might be found useful. No caustic soda would be formed, as carbonate of magnesium and sulphate of sodium would be produced.

Scorodite, or ferric arsenate, is not a rare mineral. It occurs often in conjunction with limonite, a hydrated ferric oxide. Probably these minerals, when found together, are the result of the oxidation of arsenical pyrite. Scorodite has the formula $FeAsO_4 + 2H_2O$. According to 'Comey's Dictionary of Chemical Solubilities,' it is insoluble in water, but soluble in weak hydrochloric acid. SO_2 in solution also acts on it. When a finely ground mineral containing scorodite is left in contact with a weak caustic soda solution, arsenate of sodium is slowly formed. Cyanide of sodium or potassium acts in the same way, this accounts for the slow and continuous decomposition of cyanide solutions when treating ores containing arsenates. The solution becomes charged with arsenates which are reduced in the zinc-box, and the precipitated gold carries arsenic, which is not a very pleasant thing to have in the sulphuric acid treatment, as arseniuretted hydrogen, a poisonous gas, is evolved.

I have made many experiments in order to discover a method of overcoming the injurious action of arsenates on cyanide solutions, and have devised the following method: A solution of arsenate of sodium and calcium hydrate gives the following reaction:

$$2Na_3AsO_4 + 3CaO + 3H_2O = Ca_3(AsO_4)_2 + 6NaOH$$

The calcium arsenate is insoluble, and the sodium hydrate is regenerated. I ascertained by a series of experiments that, using an excess of lime and a small quantity of sodium hydrate, all the arsenate that it was possible to dissolve was taken up by the sodium hydrate, then precipitated as calcium arsenate, no arsenic being left in the solution. The cyanide of sodium was unaffected by this reaction. The method I used in order to determine the amount of lime required was as follows: 200 gm. of finely powdered ore was shaken up every now and then, for seven or eight hours, with 200 c.c. of 1% NaOH solution, the amount of NaOH used up being determined in the usual way, and for every pound of NaOH required per ton, 0.7 lb. of CaO is required in actual practice, but of course some sodium hydrate must be present in addition to act as a solvent or carrier.

For one sample of ore I treated, which contained scorodite, 13 lb. of NaOH was required per ton of ore; by this sample method I prevented the abnormal decomposition of the cyanide, and also prevented arsenic going into solution; and by regultaing the proper proportion of lime, enough was left in solution to cause perfect settlement of the slime. It will be evident that ores containing soluble sulphates or arsenates may require a large amount of lime per ton in order to settle the slime; in the case of arsenates, if there is no sodium hydrate added, the sodium in the cyanide will effect solution of arsenate, and if only a small amount of lime is used, every trace of lime will be precipitated as an arsenate of calcium, and the cyanide of sodium will be decomposed. Roasted concentrate originally containing $MgSO_4$ and $CaSO_4$ still contains these salts after roasting, and an arsenical concentrate after roasting will possibly contain some ferric arsenate.

HEATING CYANIDE SOLUTIONS

(January 27, 1912)

The Editor:

Sir—In your issue of January 6, under heading of 'Heating Solutions,' Alfred James in his 'Progress in Cyanidation of Gold and Silver Ores During 1911,' expressed a desire for data on benefit of heat on extraction. The following tables of comparison, showing extraction on Belmont ore, with different degrees of temperature, may be of interest. The benefit shown has been borne out to our entire satisfaction in practice:

I. ORE Au 0.05 oz., Ag 18.2 oz., TOTAL VALUE $10.15

Agitator.	Agitation, Hours.	Temp.	Tailing. Au, oz.	Ag, oz.	Val.
Belmont	60	Cold 60°	0.015	3.1	$1.86
"	60	Warm 90°	0.01	1.7	1.06
Pachuca	60	Cold 60°	0.02	3.8	2.32
"	60	Warm 90°	0.015	2.1	1.36
Bottle	60	Cold 64°	0.02	3.4	2.12
"	60	87°	0.01	2.5	1.49
"	60	109°	0.01	1.7	1.06

II. ORE Au 0.05 oz., Ag 20.6 oz., TOTAL VALUE $11.30

Agitator.	Agitation, Hours.	Temp.	Tailing. Au, oz.	Ag, oz.	Val.
Belmont	48	+90°	No sample taken.		
Pachuca	48	+90°	0.015	2.6	$1.61
"	48	60°	0.02	4.5	2.67
Belmont	69	+90°	0.01	1.7	1.06
Pachuca	69	+90°	0.015	2.1	1.36
"	69	60°	0.02	3.8	2.32

Material agitated in each of the above tests was taken from collectors and contained the gold and silver remaining after concentration and that dissolved in crushing and conveying at head of mill.

To maintain a heat of 90° in the agitators, boilers are run two 8-hr. shifts during summer and three 8-hr. shifts during winter, being fired with crude oil at a cost of $2.05 per barrel at the mill, making cost per ton of ore treated $0.161 for past nine months. Degrees of heat are marked +90 to show they were kept as near 90° as possible, varying from 90 to 95 degrees.

A. H. JONES.

Tonopah, Nevada, January 17.

(February 24, 1912)

The Editor:

Sir—With reference to the benefits derived from slime treatment with heated solution, the one probably least considered is increased filter capacity. I offer the following record of results obtained with two Just laboratory filters run simultaneously, one cold and one at about 100°F., at constant vacuum on identical pulp. The wash solution in each case was cold.

A consideration of the above results will show that in preheating pulp going to the other filters (in this case a very difficult natural slime) the loading period was reduced 60%, the permeability of the cake to wash solutions increased and more undissolved gold put into solution during both loading and washing periods.

TEST No. 1

		Solution Displaced.				Cake.	
	Time, min.	c.c.	Value per ton.	KCN.	P.A.	Dry Wt., gm.	Moisture, %
Loading A at 70°F	30	1570	$1.86	1.3	0.5	825	36½
Washing A with cold water	60	785	1.45	1.1	0.4		
Loading B at 100°F	20	1570	2.08	1.3	0.5	810	34
Washing B with cold water	100	1435	0.92	0.55	0.25		

Difference by fire assay in tailing value in favor of heating pulp over loading cold on same length cycle, $0.28 per ton.

TEST No. 2

		Solution Displaced.			Cake.	
	Time, min.	c.c.	Value per ton.	KCN.	Dry Wt., gm.	Moisture, %
Loading A at 70°F	45	1635	$1.36	1.05	915	41
Washing A with cold water	50	830	1.20			
Loading B at 100°F	15	1850	1.36	1.05	885	38
Washing B with cold water	80	2460	0.42			

Difference by fire assay in tailing value in favor of heating pulp over loading cold on same length cycle, $0.28 per ton.

On the other hand, by preheating, the cycle could have been reduced from 150 minutes to 110 minutes for the same efficiency as when loading cold, increasing the filter capacity 27%. In the mill for which three tests were run the total tonnage could thus have been increased 21%, the capacity of the filter plant being the limiting factor in tonnage treated.

NOEL CUNNINGHAM.

Millers, Nevada, February 3.

By M. W. VON BERNEWITZ

(December 28, 1912)

At Tonopah crushing is done in weak and warm (from 50 to 80°F.) cyanide solutions, so the ore is in contact with solution from the stamps to filtration. This is necessary as well as the heating, which, although somewhat expensive, quickens the solution and accelerates the dissolving action. Solutions are usually heated to about 95°, and in one case 120°, by live steam introduced in the agitators.

The practice of using hot solutions is briefly as follows: At the new Belmont mill the temperature at the stamps is from 60 to 70°F., and at the Pachuca agitators exhaust steam from the mill air-compressor is fed in, increasing it from 90 to 100°. In the *Mining and Scientific Press* of January 27, 1912, A. H. Jones, metallurgist at this plant, gave some valuable data on this subject. On an ore carrying 0.05 oz., of gold and 18.2 oz. of silver per ton, 60 hours' agitation with both 60 and 90° solutions, the tailing averaged 0.0175 and 3.45, and 0.0125 and 1.90 oz. respectively. Tests on 48 and 69 hours at similar temperatures gave as marked results. Besides the effect on extraction, the hot solutions flowing through the mill kept the whole place at a good working temperature. At the Montana-Tonopah, ore is crushed in 50 to 60° solution, which is increased to 110° at the Hendryx agitators by live steam. It is found also that the heat aids settling. There is a marked decrease in extraction without hot solutions.

The MacNamara mill recently had experience with cold solutions, owing to an enforced shut-down for two days. The Trent agitators are usually kept at from 115 to 120° by live steam and it was found that it took several days to heat everything again, in the meantime the time of agitation had to be increased and extraction fell off considerably. Heat is necessary in the summer, but less steam is used. The cost is about 30c. per ton treated. The extension ore is crushed in 80° solution, and live steam is added to the Trent agitators as soon as possible, making the temperature up to 120°. It was found that this was better than 90° and extraction has improved 1.5 to 2% during the past few months, it being 94.5% at present. About 2100 tons of solution is circulating in the mill, and

it takes 7 days to heat this if it should get cold, meanwhile extraction falls off.

Cost of heating is 18c. per ton. It has been found at the Belmont mill, at Millers, that in passing through a tube-mill the temperature increases, presumably by the grinding action of the pebbles, mill liners, and ore particles. A test taken while I was there showed feed temperature at 65°, and discharge 70°. In the *Mining and Scientific Press* of February 24, 1912, Noel Cunningham, at Millers, contributed the results of some experiments, proving that laboratory work had shown greatly improved results from hot solution. At another plant treating Tonopah ore, crushing is done in 76 to 80° solution, increased to 95° in the agitators by live steam in coils. Recent tests showed a saving of 24 to 32c. at a cost of 11c. per ton. The same temperature is kept up in summer and winter. At the Mexican mill, Virginia City, solution is heated to about 96°, at a cost of 12c. per ton, results being improved by this system, the average extraction being 92 per cent.

It is to be hoped that millmen in Cobalt, Mexico, and Waihi will give their experience with heating solutions. When I was with the Waihi company in 1898, heating was tried, but I kept no data. It may be interesting to mention that, at Kalgoorlie, treating an ore containing practically no silver, some argument was raised as to the benefit of hot solutions in treatment. The heating there is not intentional, as at Tonopah, but comes about through the hot roasted ore being mixed with solution, bringing it up to nearly 200°. The discussion resolved into whether by cooling prior to mixing there would be less consumption of cyanide, and less trouble with sulphates being deposited in launders, pipes, and pans, or whether there were benefits derived from the resulting hot pulp. The Associated, Associated Northern, and Kalgurli, more particularly, found that there was no appreciable decomposition, and up to 50% of the gold was dissolved at that point; while the Great Boulder, Perseverance, and South Kalgurli preferred to cool the ore before mixing.

LUBRICATING OILS AND WARM SOLUTIONS IN CYANIDE PRACTICE

(January 18, 1913)

The Editor:

Sir—During the hot summer months on the desert it usually requires some extra effort toward conserving the limited water supply in order to meet the requirements of a milling plant. At the King of Arizona mine an endeavor was made to meet this condition by laying drains from the boilers, engines, pumps, and hoists to convey all waste water, and incidentally a large portion of the waste oil, to a common sump. As occasion permitted, the water from this sump was added to the working solution by pumping it as a final wash upon a leaching vat, thus leaving the oil deposited on top of the sands.

The Mexican hoist men had observed the oil floating on the vats from time to time, and seemed to me to show uncommon zeal

in the matter of allowing waste oil, oily refuse, cotton waste, etc., to slide into the sump. They had evidently concluded that this was a new departure in cyaniding and presumably had something to do with lubricating the path of the gold through the vats into the refinery. At least a protest against so much oil being thrown into the sump elicited the quick response: *"Pero aceite se necessita para el oro mas pronto no?"*

Exhaust pipes from the solution and vacuum steam pumps in the refinery were laid to the weak solution sumps and arranged to exhaust beneath the surface of the solution. By this means the mill was usually abundantly supplied with steaming warm weak solution charges. In discussing the effects of warm and hot cyanide solutions, late reference books on cyanide practice seem inclined to disparage any benefits to be derived from such procedure. Clennell[1] says: "In some cases it has been found that higher extractions are obtained by the use of hot solutions at a temperature of say 100 to 130°F. and provision is sometimes made for heating solution or slime pulp by the injection of steam. In most cases it is doubtful whether the improved extraction outweighs the increased cost." H. W. MacFarren[2] adds: "In practice it has generally been impossible to note any difference between the normal extraction and that made by heating the ore and solution, or that obtained during the heat of summer or the frigid weather of winter."

I wish to state here that heating the solution was done less with the expectation of directly increasing the dissolving power of the solution, than to increase the rate of percolation, thereby facilitating more through drainage, hence a better ecerated charge, and an increased number of solution charges within the treatment time allowed. Each vat was given a 7 days solution treatment. A marked decrease in the percolation rate was noticeable toward the close of the treatment period, due to packing of the sand. This was especially troublesome in the vats nearest the mill, but was largely compensated by using warm solutions toward the end of the treatment. As the mill was treating 200 tons daily a saving of 20 to 30c. per ton by utilizing waste heat proved a profitable innovation amounting to a considerable sum at the close of a month's run. While it is possible there may have been a slight loss of cyanide due to discharging the hot oily steam into the solution, such losses were not detected.

Regarding the effect of cyanide upon oils, Clennell[3] says: "Cyanide is also a solvent of oily and fatty matters, but probably only by the virtue of the free alkali it may contain." It is of interest here to note also the effect of an electrolyzed cyanide solution upon oils as observed by J. H. Aldrich, Jr.[4]; "Grease in the ore or on the surface seems to dissolve very rapidly in the treated solution and

[1]Clennell, 'The Cyanide Hand Book,' p. 239.
[2]MacFarren, H. W., 'Cyanide Practice,' p. 15.
[3]Clennell, 'The Cyanide Hand Book,' p. 184.
[4]*Trans.* Amer. Inst. Min. Eng. Vol. XLII, p. 746.

slowly in the untreated solution. We tarred our tanks inside and coated them with black oil outside, and more or less grease was frequently floating upon the surface of the solution, where the effect was noticed."

Bottle tests were prepared as follows, and the solutions covered with oil and occasionally agitated:

	I. Cylinder oil, per cent.	II. Engine oil, per cent.
Free KCN	0.20	0.20
Total KCN	0.20	0.28
Protective alkalinity	0.08	0.05
After 24 Hours		
Free KCN	0.20	0.20
Total KCN	0.20	0.26
Protective alkalinity	0.08	0.07
After 48 Hours		
Free KCN	0.20	0.20
Total KCN	0.20	0.26
Protective alkalinity	0.10	0.08

A final titration in 96 hours gave similar results, with a still further increasing in alkalinity. In two other tests from different lots of oils quite a copious white precipitate formed, but the general results did not in the main differ from those given.

J. E. CLARK.

Riverside, California, December 9, 1912.

COPPER AND SULPHUR IN CYANIDE SOLUTION

By Will H. Coghill

(August 17, 1912)

In contradistinction to a process* in which the ready solubility of copper in cyanide solution is an advantage in the extraction of zinc sulphide from the ore, I wish to show a case in which the ready solubility of copper in cyanide makes the commercial extraction of silver from the ore a difficult problem. The results relate to two similar samples which were tested simultaneously. They will be designated as samples A and B, assaying 36.27 and 26.75 oz. silver, respectively. There was less than 1% sulphide minerals and the gangue was a silicate rock. Galena, sphalerite, pyrite, and rhodocrosite were identified. The sulphides contained nearly all the silver; in fact, it was estimated that if were possible to make a concentration consisting wholly of sulphides it would assay more than 2000 oz. per ton.

After a number of tests which showed that something in these samples was possessed with an inordinate desire to consume cyanide it was decided to make a long-period air-agitation test, changing the solution frequently, thus preventing fouling and to test the pregnant solution for base metals. The quantities of reagents consumed and silver dissolved are shown in the appended Table 1.

*McGregor. *Eng. & Min. Jour.*, Dec. 2, 1911, p. 1080.

TABLE I

AIR AGITATION

No.	Amount taken in A. T. Ore.	Sol.	Length of period in hours	Total time	Strength of solution. Before. KCN	Before. CAO	After. KCN	After. CaO	Consumpt'n KCN per ton ore	Total con. KCN per ton ore	Silver recovered, %	Total Silver recovered, %	
									Sample A—Slime				
22	2	8	6	6	2.1	2.4	1.1	0.2	4.0	4.0	24.2	24.2	
23		8	12	18	1.9	2.2	1.5	0.7	1.6	5.6	8.1	32.3	
24		8	24	42	1.9	2.2	1.3	1.0	2.4	8.0	16.3	48.6	Added 6 lb, lime per ton ore.
25		8	24	66	1.9	2.2	1.5	0.3	1.6	9.6	10.1	58.7	
26		8	24	90	1.9	2.2	1.6	0.4	1.2	10.8	7.0	65.7	Tailing assay, 8.60 oz. per ton, 76.3% extraction.
27	2	8	6	6	4.1	2.4	2.5	0.4	6.4	6.4	33.4	33.4	
28		8	12	18	3.7	2.2	3.4	0.6	1.2	7.6	14.6	48.0	
29		8	24	42	3.7	2.2	3.2	1.1	2.0	9.6	17.4	65.4	Added 6 lb. lime per ton ore.
30		8	24	66	3.7	2.2	3.3	0.3	1.6	11.2	7.8	73.2	
31		8	24	90	3.7	2.2	3.6	0.3	0.4	11.6	5.2	78.4	Tailing assay, 3.24 oz. per ton, 91.2% extraction.
									Sample B—Slime				
32	2	8	6	6	2.1	2.4	1.4	0.4	2.8	2.8	22.8	22.8	
33		8	12	18	1.9	2.2	1.6	0.7	1.2	4.0	8.8	31.6	
34		8	24	42	1.9	2.2	1.6	1.1	1.2	5.2	16.2	47.8	Added 6 lb. lime per ton ore.
35		8	24	66	1.9	2.2	1.6	0.5	1.2	6.4	11.8	59.6	
36		8	24	90	1.9	2.2	1.7	0.5	0.8	7.2	6.5	66.1	Tailing assay. 5.00 oz, per ton, 81.4% extraction.
37	2	8	6	6	4.1	2.4	3.3	0.4	3.2	3.2	29.2	29.2	
38		8	12	18	3.7	2.2	3.3	0.8	1.6	4.8	13.1	42.3	
39		8	24	42	3.7	2.2	3.1	1.4	2.4	7.2	15.2	57.5	Added 6 lb. lime per ton ore.
40		8	24	66	3.7	2.2	3.2	0.3	2.0	9.2	9.5	67.0	
41		8	24	90	3.7	2.2	3.2	0.3	2.0	11.2	7.9	74.9	Tailing assay, 3.32 oz. per ton, 87.7% extraction.

Note that the consumption of KCN per ton of ore during 90 hours agitation ranged from 7.2 to 11.6 lb. Note also that the greatest cyanide consumption was during the first six hours and a greater percentage of the silver was dissolved during this period than during any subsequent. The large discrepancy between 'total percentage Ag recovered' and percentage extracted as determined by assay of tailing is to be accounted for by the unavoidable losses during transfer of solutions. The assay of tailing is correct and indicates the percentage extracted; however, the figures under 'percentage silver recovered give a good idea of the rate of dissolution.

In 31, for instance, it is noted that only 5.2% of the silver was dissolved and this is easily explained by the fact that most of the silver having been previously extracted, there was little left to dissolve. Cannot the decreased consumption of cyanide be explained along the same lines, that is, the cyanide is in very small quantities and is practically all consumed during the first 90 hours?

Table II throws some light on this question. It is the result of the determination of the amount of copper in the samples and in the solutions.

TABLE II

No.	Amount taken in A.T. Ore.	Solution. 2 lb.	Solution. 4 lb	Copper in ore, %.	Copper in sample mg.	Copper dissolved, mg.	Copper dissolved, %	Total copper dissolved, %	Total silver dissolved, %
22-26	2	40		0.29	168	17.6	0.03	10.0	76.3
27-31	2		40	0.29	169	25.0	0.04	14.8	91.2
32-36	2	40		0.21	122	15.0	0.03	12.3	81.4
37-41	2		40	0.21	122	29.1	0.05	23.9	87.7

The analysis of the samples shows a quantity of copper (0.29 and 0.21%) that would be considered negligible in most cases, but not so in this; for in 37-41 23.9% of this copper was dissolved. It seems, then, that the suspicion that the cyanide was in small quantities and finally spent itself was correct, though the percentage of total copper dissolved is barely comparable with that of the total silver dissolved.

In order to secure more pregnant solution for further tests, a grinding in cyanide test was made, lasting 48 hr. The tests were each aerated twice. Table III shows the status regarding dissolved copper and silver. The results are nearly identical with those shown in Table II and are here recorded to show that the amount of copper dissolved is independent of quantity of solution and method of treatment—that is, whether by air-agitation or grinding in cyanide.

TABLE III

GRINDING IN 4-LB. CYANIDE SOLUTION

Name	Amount taken A. T. Ore.	Sol.	Copper in ore, %	Copper in Sample, mg.	Copper dissolved mg.	Copper dissolved %	Total copper dissolved, %	Total silver dissolved, %
Sample A	2	8	0.29	169	23.7	0.04	14.0	76.3
Sample B	2	8	0.21	122	31.6	0.05	26.0	83.3

Table IV, which follows, portrays, I think fairly accurately, the conduct of the cyanide in the test shown in Table III.

TABLE IV

	Sample A.		Sample B.	
KCN as:	Lb. per ton solution.	Per cent.	Lb. per ton solution	Per cent.
Free cyanide	1.30	32.9	1.60	40.6
KCNS	1.17	29.6	0.91	23.0
$KAgCN_2$	0.57	14.4	0.46	11.6
$4KCN.Cu_2CN_2$	0.73	18.5	0.97	24.6
Error	0.18	4.6	0.01	0.2
Total	3.95	100.0	3.95	100.0

The amount of free cyanide remaining was determined in the usual way, with silver nitrate and potassium iodide indicator. The table shows that only 32.9% and 40.6%, respectively, of the cyanide added remained as free cyanide. The KCN as KCNS was determined by the permanganate method for estimation of thiocyanates as outlined by Clennell in 'Chemistry of Cyanide Solutions.' The KCN as $KAgCN_2$ is an estimation based on the amount of silver dissolved. The copper salt was calculated in the same way. Inspection of the 'error' column shows that all but a very small amount of the KCN is accounted for; but after all, one cannot be sure. And here I wish to point to a paucity of published information. During each of the 52 weeks of the year there are several articles published in our mining magazines alleging to describe cyanide processes. They always begin by stating that the ore is run onto grizzlies, and near the end state the strength of cyanide solution, seldom referring to cyanide losses. If cyanide loss is mentioned, it may possibly be subdivided into physical and chemical, but the reader will have to search a long while to find out what constitutes the chemical losses. If the cyanide chemist does not investigate and find the cause of chemical losses, he is either overworked or negligent. If he investigates and does not publish his results, he is not doing his duty to fellow-workers.

In the case of the samples under discussion, it would certainly be unscientific to dope the solution with this and that before finding what salts are present. Clennell has indeed given us two excellent books to aid in such determinations. However, there are yet some uncertainties. I refer to the copper compound. I will quote here from Clennell's 'Cyanide Handbook,' page 80:

"The compound $4KCN.Cu_2CN_2$ is the form in which copper, according to Virgoe, normally exists in cyanide solutions, though Sulman considers that it is more frequently present as sulphocyanide, $Cu_2(SCN)_2$, dissolved in KCN. * * * The reaction (in the presence of an excess of alkali) which, according to the writer's experiments, seems most in accordance with the facts, is as follows:

$$2Cu(OH)_2 + 7KCN = 4KCN.\,Cu_2CN_2 + 2KOH + KCNO + H_2O$$

This indicates a consumption of 3.6 parts KCN for every part of Cu dissolved."

The calculations in Table IV were based on the above statement. Referring to page 112 in the same book, I take the liberty to multiply (a) and (c) by 2 and (d) by 3 for reasons that will be obvious later.

"The natural sulphides of copper are much less readily acted on by cyanides than are the carbonates, or than artificially prepared sulphides, but they gradually dissolve, probably with formation finally of cupric thiocyanate, by some such series of reactions as follows:

$$\text{(a)}\quad 4Cu_2S + 8KCN + 4H_2O + 2O_2 = 2Cu_2\,(CNS)_2 + 2Cu_2CN_2 + 8KOH$$

The insoluble cuprous thiocyanate and cyanide readily dissolves in excess KCN.

$$\text{(b)}\quad 2Cu_2(CNS)_2 + 8KCN + H_2O + O = 2Cu(CNS)_2 + Cu_2CN_2.6KCN + 2KOH$$

$$\text{(c)}\quad 2Cu_2CN_2 + 12KCN = 2(Cu_2CN_2.6KCN)$$

Since the solutions almost invariably contain an excess of alkaline thiocyanate, the further reaction is probably as follows:

$$\text{(d)}\quad 3(Cu_2CN_2.6KCN) + 12KCNS + 3H_2O + 3O = 6Cu(CNS)_2 + 24KCN + 6KOH."$$

Upon adding these equations, the following equation results:

$$4Cu_2S + 12KCNS + 4KCN + 8H_2O + 8O = 8Cu(CNS)_2 + 16KOH$$

Here it is seen that 8 times 64 parts of copper consumes 16 times 65 parts of KCN, or one part of copper consumes two parts of KCN, which is a much more favorable ratio than the one previously cited, and has further in its favor the decomposition of some of the KCNS that naturally forms in the presence of sulphides.

Using the data which I have derived from Clennell's four equations, Table IV may be recalculated as below:

TABLE V

KCN as:	Sample *A*. Lb. per ton solution.	Per cent.	Sample *B*. Lb. per ton solution	Per cent.
Free cyanide	1.30	32.9	1.60	40.6
KCNS	0.76	19.2	0.37	9.4
$KAgCN_2$	0.57	14.4	0.46	11.6
$Cu(CNS)_2$	0.41	10.4	0.54	13.7
Unaccounted for	0.91	23.1	0.98	24.7
Total	3.95	100.0	3.95	100.0

Here two thiocyanates have to be reckoned with instead of one. Taking into account the amount of copper in solution, the KCN as cupric thiocyanate is first calculated and all thiocyanate left over is reckoned as KCNS. The amounts of KCN as Cu $(CNS)_2$ is 0.41 and 0.54, respectively. An inspection of the formulæ of these two thiocyanates show that the results of titration are the same whether the sulphur is in KCNS or Cu $(CNS)_2$; therefore, to find the amount of KCN as KCNS (left over after satisfying the copper), it is necessary only to take the results from the KCNS column in Table IV, which are 1.17 and 0.91, and substract from them 0.41 and 0.54, respectively. This gives 0.76 and 0.37. In other words, the quantities under $4KCN.Cu_2CN_2$ vanish, and there are two thiocyanates which are not distinguishable by titration with permanganate. The 'error' is thus increased by the amount of KCN as $4KCN.Cu_2CN_2$ which has been eliminated, and it seems best to call it 'unaccounted for.'

It is worthy of note that there is less difference in the percentages 'unaccounted for' than there is in the 'error'; these being 4.6% and 0.2% in Table IV and 23.1% and 24.7% in Table V. I cannot explain away such a large percentage under 'unaccounted for'; for, as I have shown, our authority has not stated positively how the copper occurs. The grinding was done in an Abbé sample jar, and since the protective alkalinity did not get below 0.3, certainly no HCN was formed.

A reference to alkalinity suggests another phase of the investigation that may be of interest. It relates to the acidity of these samples. Naturally the first test made was a half-hour agitation in lime water, to determine acidity. This was found to be sufficient to neutralize 3.6 lb. CaO in each case. It soon developed that this test had failed to reveal the true conditions; that the indicated acidity depended upon the time of treatment and state of comminuation. This is shown in Table VI.

The 'dry crushed' was crushed dry in an Abbé jar mill, and was so fine that 85% passed 150-mesh; while the 'slime' was ground wet in the same apparatus until all passed 150-mesh. An attempt was made to find the cause of this accumulating acidity, but lack of time

TABLE VI

ACIDITY TEST

Sample A—Agitation

	Amount in A. T.		Strength of solution.			Lime consumed per ton of ore.	
	Ore......	Solution....	Before......	After 2 hr...	After 24 hr..	After 2 hr...	After 24 hr.
Dry crushed	1	6	2.0	1.2	1.1	4.8	5.4
Slime	1	6	2.0	0.9	0.6	6.6	8.4

has prevented a thorough investigation. I will say, however, that suspicion rested upon the copper. Much has been written lately about the ready oxidizability of copper; for instance, Durant[2] says: "There are few published records showing that unoxidized copper ores are actually leached in place on a working scale, but such is actually the case, and the fact is that it has been carried on, day in and day out, for over 60 years at a mine barely 300 miles from London."

Then again Bushnell[3] says, regarding the precipitation of copper from mine waters at Butte: "Some of the operators are so fortunate in the situation of their plants that they can cause the water to flow through the tailing heaps of abandoned plants, thereby materially increasing the copper contents of the water."

The air-agitator was therefore put to work and ran five days on a pulp consisting of slime and water. The solution, at the end of this period, instead of revealing copper sulphate and acid, showed that acidity was absolutely *nil*—another theory exploded. Since the copper mineral in this slimed material resists oxidation, while archalcocite, Cu_2S, is known to oxidize easily, it seems probable that the copper does not occur as Cu_2S but in some mineral such as argentiferous tetrahedrite, for instance.

Conclusions

1. Copper, though in very small quantities in the ore, may be a detriment to cyanidation.

2. Sulphur, though but a small amount of sulphide be present, may form an appreciable amount of thiocyanates.

3. In order to find how much cyanide has been consumed by copper, it does not suffice to make analysis for copper, but the com-

[2] *Eng. & Min. Jour.*, Nov. 11, 1911.
[3] *Mining and Scientific Press,* Nov .18, 1911.

position of the copper salt must be known. This ratio may be anything between 3.6 to 1 and 2 to 1.

4. For a given amount of copper entering the solution, it is more desirable, from the standpoint of cyanide consumption, for it to occur as a sulphide than as an oxide.

5. The 30-minute agitation test in lime water does not accurately indicate the acidity of an ore, since the indicated acidity depends upon length of period of treatment and state of comminution of the sample.

6. More data should be collected on the subject of chemical losses. This must largely emanate from the ore-testing laboratory for the simple reason that a chemical loss of much magnitude prohibits treatment on a commercial scale. If interest should be aroused in this subject, it would be but a short time until there would be enough published data to be of aid in devising schemes to overcome chemical losses, thus making many ores available to cyanidation which cannot be so treated at present. The average client is generally reluctant to pay for purely scientific investigation, but the cyanide chemist who will not make a few tests purely for the fun of it, has failed to see the best that is in his profession.

ACTION OF OXIDIZING AGENTS ON THE VELOCITY OF SOLUTION OF GOLD IN POTASSIUM CYANIDE

(October 19, 1912)

By suspending a piece of Au for 10 minutes in a 0.2 N solution of KCN saturated with air and determining the loss of weight both in absence and in presence of other substances, the following results were obtained by Ya. Mikhailenko and M. I. Meshcheryakov, as given in the *Jour.* Russ. Phys. Chem. Coc., 44, 567-70: Introduction of H ions gradually diminishes the velocity of solution, an excess of them entirely arresting the action. OH ions do not accelerate the solution, and an excess of them has a retarding influence. In neutral solution, the follownig substances have no influence on the velocity: quinine, $Na_2SnO_3.3H_2$, $KBrO_3$, KIO_3, $KClO_3$, $Hg(CN)_2$, $CuCl_2$, $2H_2O$, while $KClO_4$, $KMnO_4$, NH_4SO_4, Na_2O_2, KSO_4, $NaSO_4$, Br, $K_3Fe(CN)_6$, and KCO_3 accelerate the solution. Taking the accelerating influence of $KClO_4$ (which is not large) as unit, the influence of the following substances can be represented as follows: KIO_4 or KCO_3 2, NH_4SO_4 3, KSO_4, $NaSO_4$, and Na_2O_2 4, $K_3Fe(CN)_6$ 5. The accelerating influence of these substances increases with their concentrations, but only up to a certain limit, above which they begin to exercise a retarding effect. This maximum is 0.02 of an equivalent (*E*) for Na_2O_4 and NH_4SO_4, 0.04 *E* for $K_3Fe(CN)_6$, $NaSO_4$ and KIO_4, 0.1 *E* for KSO_4, and 0.2 *E* for KCO_3. The minimum amount capable of accelerating the reaction is in most cases 0.004 *E*. A variation of the concentration of the KCN solution from 0.2 to 0.05 *N* had no influence on the maximum effect

of the oxidizing agent $K_3Fe(CN)_6$. The sum of the influences of several oxidizing agents is less than the maximum of the strongest of them. Addition of NaCl, $CuCl_2$, $Hg(CN)_2$, and $CoCl_2$ has either no influence on the velocity or retards it.

SEPARATION OF BASE METALS IN CYANIDE SOLUTION FOR QUANTITATIVE DETERMINATION

By P. L. Guppy and Douglas Waterman

(November 9, 1912)

In the treatment of silver sulphide ores by the cyanide process, free alkaline sulphides are carried into solution. Lead acetate is often used as a precipitant for these free sulphides. Any excess of lead acetate over the amount required to accomplish this purpose is undesirable, as it is carried to the zinc-boxes in the form of sodium plumbate and there deposited with the silver. This entails a needless waste, not only of the acetate, but of the zinc as well, which is consumed in the process of replacement. An excess of lead acetate led to the production of a base bullion. It is highly desirable, therefore, to know if lead is present in excess. The amount of zinc and copper in solution is also of importance at times. By the following scheme of analysis these and other base metals may be recognized and separated for quantitative determination by any of the well known methods.

Partly fill an Erlenmeyer flask with 500 c.c. of the cyanide solution to be analyzed. Place a small glass funnel, with stem broken off, in the neck of the flask, which will effectually prevent the formation of excessive quantities of steam thus avoiding all danger of the solution boiling over.

I. Add 15 c.c. concentrated HCl, salts of copper, a portion of the silver, and ferro-cyanides of the heavier metals will be precipitated. Evaporate to a volume of 100 c.c. and allow the solution to cool. Add 300 c.c. of bromine water and boil until all the bromine is expelled. This should bring the ferro-cyanides of zinc, by oxidation, into soluble form.

Dilute to 400 c.c. with boiling water, and pass a rapid current of hydrogen sulphide through the solution until precipitation is complete. On withdrawing the current of H_2S the precipitate should settle rapidly, leaving the solution clear, showing that precipitation is complete. Filter through an 11-cm. paper and wash the precipitate with H_2S water. The result then is:

(a) Precipitate containing silver, gold, copper, and lead; antimony an arsenic if present.
(b) Filtrate containing zinc, manganese, and other metals not precipitated by H_2S.

II. If it is desired to test for antimony and arsenic, the funnel is inverted over an 8-oz. beaker, into which the precipitate is

washed with a fine jet from a wash-bottle; 15 or 20 c.c. of sodium sulphide is added, and the contents of the beaker digested for 10 or 15 min. to bring the antimony and arsenic into solution.

Filter through the same filter-paper into a clean beaker, wash with dilute sodium sulphide, and finally with a little warm water. On acidifying the filtrate with HCl the presence of antimony or arsenic may be readily detected; antimony giving an orange-colored precipitate, and arsenic a yellow.

These substances may be determined quantitatively by any of the usual means, but an excellent method for determining small quantities of antimony and arsenic, with full explanations of necessary precautions, will be found in the first edition of Fresenius' 'Quantitative Analysis.'

III. Wash the bulk of the precipitate (a) into a casserole with hot water. Remove the filter-paper and spread over the convex surface of a watch-glass. The adhering precipitate is dissolved with hot dilute nitric acid and added to the contents of the casserole. (Care should be taken that none of the filter-paper is allowed to fall into the casserole.) Cover the casserole and heat until precipitate is dissolved. Add 20 c.c. concentrated H_2SO_4 and evaporate on hot plate until copious white fumes are given off. When cool, rinse the contents of the casserole with a little cold water into a clean beaker. The presence of lead may be readily detected by the white granular appearance of the precipitate in the bottom of the beaker, and is easily distinguished from either the chloride or the sulphate of silver, which may also have been precipitated.

IV. Filter and wash with sulphuric acid diluted 4 to 1.

(c) Precipitate containing lead (if present) and a small quantity of silver.
(d) Filtrate containing copper and nearly all of the silver.

To the filtrate (d) add just enough sodium chloride solution to precipitate the silver, which is then removed by filtration. The copper in solution is precipitated on aluminum foil. For quantitative determination of both the lead and copper, reference should be made to Low's 'Technical Methods of Ore Analysis,' fifth edition, pp. 144 and 84.

V. To the filtrate (b) containing zinc and manganese add 20 c.c. more HCl and boil to expel the hydrogen sulphide. Add a few drops of nitric acid and heat for five minutes. Add 5 gm. ammonium chloride, and then ammonia until the solution is alkaline.

(e) Precipitate containing iron and aluminum.
(f) Filtrate containing zinc and manganese.

The iron in precipitate (e) may be determined quantitatively and expressed in terms of ferro-cyanide if desired.

VI. To the filtrate (f) add a slight excess of ammonium sulphide to precipitate the zinc and manganese. Nickel and cobalt would also be precipitated at this point, but as they are of rare occurrence their treatment will not be considered.

Dissolve the precipitate in hot dilute HCl and boil until the H_2S is completely expelled. Cool and add ammonia in slight excess. Add bromine water until the solution is well colored by this reagent, and boil. Manganese will be precipitated and may be determined by the method given in the first edition of Fresenius' 'Quantitative Analysis.'

The zinc will remain in the filtrate. Acidify with HCl, add 3 c.c. in excess, and evaporate down to 250 c.c. It may then be determined as outlined in Low's 'Technical Methods of Ore Analysis.'

THE GITSHAM PROCESS

(November 23, 1912)

This new variation of the cyanide process is attracting a good deal of attention in Australia. The *Journal* of the Chamber of Mines of Victoria, makes the following statement regarding it: A subject of interest to the mining community is the process of the Gitsham Gold Extraction Co. It is claimed for this process that it will extract gold from refractory materials where the cyanide process has wholly or partly failed. Antimony and copper ores are readily treated by the solvent, which is a weak solution of hydrocyanic acid. The company has treated at a profit about 4000 tons of material at Costerfield, Burke's Flat, and Clonbianne, which has been a stumbling block to many cyaniders. Experimental work on pyritic tailing from various parts of the Commonwealth has shown excellent results, a notable case being tailing from Randall's, Western Australia, which needs no comment. Unlike the cyanide solution, the Gitsham solvent is a weak acid, and it is claimed that owing to this fact material can be treated by leaching which otherwise would require agitation. The percentage of regeneration of the solutions, it is stated, is high, and averages from 40 to 50% of the chemicals used. The company claims that, given on ore partly amenable to the cyanide treatment, their process will give an increased extraction at a lower cost; and even on clean material the costs of chemicals are less, the only exception being on ores containing carbonates of lime, and so far this difficulty has not been overcome. As this type of material is very rare, it is not a serious matter. The control and testing of the solvent is soon learned by the man of average intelligence. The company has been doing experimental work for the past eighteen months, and is now open to the criticism of the metallurgical world.

The following supplementary details were given in a recent issue of the Kalgoorlie *Miner*. The process consists of the use of an acid solution formed by the combination of potassium cyanide and sulphuric acid, giving rise to hydrocyanic acid. Its strength varies from 0.05 to 0.1%, and is worked on the same system as ordinary cyanide. The solutions are allowed to percolate in the ordinary way, but before passing through the extractor box they are drawn off into a solution of limewater, which regenerates the

cyanide and makes the solution ready for precipitation. Should the solutions be used in the weak state, a little cyanide is added at the head of the zinc-box and the solution brought up to about 0.08%. After passing over the zinc the solutions are again made acid and applied to the ore.

The following equations show what the inventor believes really takes place:

$$2KCN + H_2SO_4 = K_2SO_4 + 2HCN$$
$$4HCN + 2Au = 2HAu(CN)_2 + H_2$$

after the zinc-box:

$$Ca(CN)_2 + H_2SO_4 = CaSO_4 + 2HCN$$

or

$$2KCN + H_2SO_4 = K_2SO_4 + 2HCN$$

The metallugist of the company claims that all copper ores, likewise antimony, bismuth, and arsenic, are insoluble, and do not affect the solution and its selective action for the precious metal. Even the most refractory copper carbonate, with mixed oxides, has been successfully experimented on. About four thousand tons of ore, containing antimony and copper, have been treated by this process for a yield of £1600 worth of bullion, of an average value exceeding £4 per ounce.

Although very delicate, the process is easily controlled by a chemist, the important feature being the combination point of KCN and H_2SO_4. Some ores have quite enough acidity to generate with the cyanide the acid. When the latter is in excess it is removed or neutralized. Where not present, the use of sulphuric is resorted to. The inventor (Mr. Gitsham) claims to be able to treat ores which have before not been amenable to the cyanide process. Costs are from 3 to 6s. per ton on the most refractory types. The process has been patented in the principal countries of the world, and the shareholders have decided to raise the additional capital necessary to introduce it in mining fields outside the Commonwealth. With this object in view, G. Gitsham and T. H. Davies are shortly leaving for London. As far as Australia is concerned, the company announces that they are prepared to deal with mine-owners on a royalty basis.

The suggestion recently made by W. D. Williamson in a communication to the *Journal* of the Chemical, Metallurgical & Mining Society of South Africa, is of interest in this connection. Mr. Williamson states that the addition of sulphuric acid to a solution containing the double cyanide of copper will cause the following reaction:

$$Cu_2(CN)_2 2KCN + H_2SO_4 = Cu_2(CN)_2 + K_2SO_4 + HCN$$

The insoluble copper cyanide can be separated by filtration, and the filtration neutralized by a slight excess of alkali. It is obvious, however, that there is still cyanogen in the copper precipitate which ought to be recovered if possible. If the precipitate be suspended

in water, and treated with hydrogen sulphide, the following reaction occurs:

$$Cu_2(CN)_2 + H_2S = Cu_2S + 2HCN$$

The solution resulting from filtration of the mixture can then be neutralized with alkali and used as solvent for fresh ore. The two reactions can be combined by adding sodium sulphide and the requisite amount of acid to the cupriferous cyanide solution, with this reaction:

$$2KCN.Cu_2(CN)_2 + Na_2S + 2H_2SO_4 = Cu_2S + 4HCN + Na_2SO_4 + K_2SO_4$$

The precipitated copper cyanide carries down with it the precious metals and can be collected, dried, and shipped to the smelter. The clear liquor is then neutralized with a slight excess of alkali to combine with the free HCN.

An obvious drawback of both of these processes is that any attempt to neutralize an acid sulphate solution by the addition of milk of lime will cause the formation of a copious precipitate of $CaSO_4.2H_2O$, which occludes considerable quantities of cyanide and which is too fine grained to permit filtering and washing unless the solution is kept cold and allowed to stand several days during precipitation. Since the hydrate of calcium is only slightly soluble in water, being less soluble than the sulphate, any attempt to neutralize in this way would greatly dilute the cyanide solution. It might prove feasible, however, to use dried or ground slaked lime for neutralizing, drive off and collect the HCN gas, and allow the sulphate solution to run to waste, or to otherwise modify the process so as to reduce it to a better working basis. The article on the regeneration of cyanide solution by B. G. Nicholl in the *Mining and Scientific Press,* March 16, 1912 (reprinted page 31) and its discussion by R. P. Wheelock in the issue of April 6, will be of interest in this connection, as will the description of the cyanidation of antimonial tailing in the Hillgrove district of New South Wales, given in our issue of June 29, 1912.

PRECIPITATION OF GOLD AND SILVER BY CARBON

By R. K. Cowles

(December 7, 1912)

*This paper is the outcome of experiments carried on and experience gained at the Waihi-Paeroa Gold Extraction Co.'s tailing plant at Waihi, New Zealand. The plant was erected to treat accumulated tailing which had been discharged into the Ohinemuri river by the Waihi stamp-mills. The Ohinemuri river is a common sludge channel for all mining débris, tailing, and ashes from the

*From the *Proceedings* of the Australasian Institute of Mining Engineers.

power-plants, the last being the cause of occasional high residues which gave so much trouble to the management.

The tailing was dredged from the bed of the river into barges by means of a Pohle air-lift, the power being supplied by a 13 b.hp. Tangye oil-engine driving an air-compressor and a centrifugal pump. The sand was elevated from the barges by bucket-belt elevators over screens, to eliminate large particles and rubbish, into storage hoppers, and from there fed direct into four C. Judd Ltd. tube-mills. The method of treatment was fine grinding in a weak cyanide solution, agitation in Brown and McMiken tanks (Pachuca), filtration with turn-over vacuum filters, and precipitation with zinc shaving.

The comparatively high content of the residue in gold gave a great deal of trouble, and it was not until repeated experiments were carried out that the cause of the trouble was identified. While the content in silver remained stationary, the gold fluctuated greatly. To overcome this, finer grinding was tried, with poor results, except that more silver was recovered—the gold still remaining too high. It was well known that carbon would precipitate gold and silver, but its selective action was not known. When fine grinding did not give the desired result, other methods had to be sought. Knowing that there was a variable amount of half-burned coal among the tailing, it was decided to experiment with carbon.

A heavy freshet occurring at that time gave the opportunity on a large scale, as it brought down a large amount of half-burned coal. On the newly deposited tailing experiments were carried out, and it was found that silver residue, just after grinding, was normal, while the gold was slightly high. Samples were then taken every day from the B. and M. tanks and assayed, with the result that the gold content unmistakably increased while the silver content decreased until such a time when practically all the gold was precipitated, then the content of the silver residue started to rise, thus showing that the carbon had no apparent effect on the silver until nearly all the gold was precipitated. The following are the results of a few assays, showing the action of carbon on gold and silver in cyanide solutions. The samples were first taken from the overflow from the classifier, and then daily from the B. and M. tanks.

Sample	Gold, oz.	Silver, oz.
No. 1.—From overflow	0.088	0.908
B. and M. tanks:		
first day	0.098	0.658
second day	0.102	0.634
third day	0.108	0.628
fourth day	0.112	0.672
No. 2.—From overflow	0.060	0.860
B. and M. tanks:		
first day	0.068	0.672
second day	0.088	0.654
third day	0.096	0.604
fourth day	0.096	0.684

Sample	Gold, oz.	Silver, oz.
No. 3.—From overflow	0.044	0.680
B. and M. tanks:		
first day	0.062	0.620
second day	0.084	0.610
third day	0.100	0.650
fourth day	0.100	0.814

No. 4.—Experiments were then made with gold and silver solutions in which ground clinker was placed, the whole being agitated for two days, with the following result:

	Gold, oz.	Silver, oz.
Before agitation	0.046	0.642
After agitation	0.013	0.504
Precipitated	0.033	0.138

No. 5.—Unconsumed coal-dust was also placed in a gold and silver solution and agitated for two days, but no precipitation occurred.

No. 6.—A sample of half-burned coal from the river was assayed, and was found to contain 0.102 oz. of gold and 0.604 oz. of silver.

No. 7.—A sample of charcoal (in small lumps) found floating in the lime tank was assayed and found to contain 1.760 oz. gold and 1.792 oz. silver. This showed that it was imperative to eliminate the charcoal associated with lime used for settling purposes.

By comparing the time of the river freshets and the slime residue of the tailing taken at that time, it was found that the residue was higher than at any other period. From this it was concluded that fresh carbon precipitated gold and silver more freely than old.

To eliminate the carbon two 6-ft. Union vanners were installed. Though these were overtaxed, the desired result was at once obtained. Besides removing the carbon, the light river sand was also removed. The output of the plant was 80 tons per day, and that amount was run over the vanners. The vanners were driven at about 120 revolutions, with a belt travel of 6 ft. per minute, and with a fall of 8 inches in the whole length. The distributing-box was placed 18 in. farther down than is usual, while the pulp discharged from the lower side. The water distributor also was placed lower down the table. The sand was from ¼ to ⅜ in. thick as it came comparatively dry over the head, where it was washed off with a series of solution jets into a launder.

In the assays No. 1, 2, and 3 there was no apparent precipitation of silver until nearly all the gold was precipitated, but in the experiment No. 4 and assays No. 6 and 7 silver was shown as being precipitated in equal and greater quantities than gold. From this apparent contradiction it is assumed that, while the silver was being dissolved, a very small amount was being precipitated, showing that the dissolution of silver up to a certain point was quicker than the precipitation.

FIRST AID FOR CYANIDE POISONING

By W. H. KRITZER

(January 18, 1913)

Considering the large quantities of cyanide salt and solution handled in gold and silver-mining districts, and the poisonous nature of this useful compound, the number of deaths from its use are few. When a man has inhaled prussic acid gas, or swallowed some cyanide solution the poison acts quickly, and his life depends on prompt action. The gas, when pure, causes almost instant death; and when diluted with air, results in dizziness, faintness, and a depressing headache. Solutions also act quickly internally while with many men they act on the skin and produce eruptions which are painful.

If an employee has inhaled prussic acid gas proceed as follows: Dash water on the patient's face; start artificial respiration; make him inhale either a small quantity of ammonia, ether, or chlorine gas. The latter may be quickly made by sprinkling a small amount of chloride of lime on a flannel cloth moistened with acetic acid, and then holding the cloth to the nostrils of the patient. If poisoning is from swallowing cyanide solution, place the patient in a hot bath, if procurable, and apply cold water to the spine and neck, providing that no delays are permitted to intervene in carrying out previous instructions. Also, incite vomiting by tickling the back of the patient's throat with a finger or feather; by giving lukewarm water, or strong mustard and water; by using a stomach tube and hot water; or by physical means. Diluted solutions of ammonia, cobalt nitrate, peroxide of hydrogen, or freshly precipitated carbonate of iron may be given. The last mentioned is made by mixing equal parts of ferrous sulphate, and sodium carbonate, and then administering at once. The following may be also used, the apparatus consisting of one sealed bottle 30 c.c. of caustic potash; one sealed bottle containing 30 c.c. 33% solution of ferrous sulphate; and one sealed package containing oxide of magnesium. As quickly as possible, empty the contents of the two bottles and a package into a metal cup, and stir thoroughly with a metal spoon. If the patient is conscious, make him swallow the mixture at once, and then lie down a few minutes. If unconscious, place him on his back and pour the mixture down the throat in small quantities, if necessary pinching the nose in order to start swallowing. Then incite vomiting by one of the previous methods suggested. It is advisable to have a soft rubber stomach-tube having a funnel and exhaust bulb in a cabinet to be described, if the services of a doctor cannot be had promptly. It is advisable to have a physician prepare the solutions required, and prescribe the dose of each to be taken by a patient in an emergency; and to further assist in preventing mistakes, bottles of blue glass can be used for the acids, and of white glass for the alkalies.

In all cyanide plants it is important to have wood cabinets with compartments of suitable size in conspicuous, and easily accessible

parts of the building, and preferably painted red. Have the emergency orders for the handling of patients fixed to the inside of the cabinet door, and properly label each bottle and see that the contents are kept in a fresh and pure condition. The four compartments should contain (1) one large bottle of distilled water; one large metal spoon; and a one-pint metal cup for mixing the different antidotes; (2) for external use one bottle each of ammonia, ether, acetic acid, chloride of lime (bleaching powder); and a piece of flannel cloth for administering the chlorine gas; (3) for internal use one bottle each of diluted ammonia, cobalt nitrate, ferrous sulphate, and sodium carbonate, the two latter to be fresh; and (4) one sealed bottle with 30 c.c. of caustic potash, one sealed bottle with 30 c.c. of 33% solution of ferrous sulphate, and one sealed package with oxide of magnesium. Another important point, now fortunately observed at most cyanide plants, is the erection of notice-boards in different parts, warning employees and others that certain tanks and pipes contain solution and others water.

THEORY OF THE DISSOLUTION OF METALS BY CYANIDE

By J. B. Stuart

(August 6, 1910)

Not long ago the tendency of those cyaniding silver ores was to rely upon a maximum degree of dilution to accomplish the maximum extraction, believing that the rate at which the silver went into solution was accelerated thereby. Hence they tried to use as many tons of barren solution per ton of slime as possible. Later this idea gave way to better practice, and the primary reason for the large bulk of solution required was found to be merely the need of washing the dissolved metals from the charge. With the perfection of filter plants, as against decantation plants, it became apparent that washing of dissolved gold and silver by dilution was by no means as perfect in the agitation and decantation method as in its displacement by the filtration methods of washing. Then it was appreciated to what extent mechanical-agitation methods suffered, from revolution *en masse* of solution and solids. In the case of filter methods of washing, there could be no doubt that the barren solution, or water, actually displaced the metal-bearing solution, retained as moisture in the cake. On the other hand, a quantity of barren solution, or water, several times as great as that used in filter methods, failed to accomplish anything like so satisfactory a 'wash' when applied by mechanical-agitation methods. This, I take it, points very clearly to the need of maintaining as high a difference between the relative velocities of slime particles and solution as possible; and the primary object of such methods I consider is to promote the diffusion of the dissolved metal from the immediate surface of the slime particles to the general mass of solution involved in the charges.

TABLE OF SURFACE AREA OF PARTICLES

Screen No.	Screen coef.	Per cent max. surface exp.	Sq. m. sur. per 1 m. ton sp.gr. 25	Sq. m. sur. per cu. m.
10	1	3.03	960	2400
20	4	6.06	1920	4800
30	9	9.09	2880	7200
40	16	12.12	3840	9600
50	25	15.15	4800	12000
60	36	18.18	5760	14400
80	64	24.24	7680	19200
100	100	30.30	9600	24000
110	121	33.33	10560	26400
120	144	36.36	11520	28800
130	169	39.39	12480	31200
140	196	42.42	13440	33600
150	225	45.45	14400	36000
160	256	48.48	15360	38400
170	289	51.51	16320	40800
180	324	54.54	17280	43200
190	361	57.57	18240	45600
200	400	60.60	19200	48000
250	625	75.75	24000	60000
300	900	90.90	28800	72000
330	1000	100.00	31680	79200

To illustrate the increased efficiency obtained in treatment by moving the solution relatively faster or slower than the slime particles, the following comparison was made of treatment by the Brown or Pachuca tank method and that of mechanically-rotated paddle-agitation assisted by vanes on sides of the tank and one 4-in. air-lift at one side. To make the comparison a little more satisfactory, the screen tests on the charge were reduced to a single figure representing the mesh number of a uniform material which would present the same surface per ton for contact with solution, as that indicated by the screen tests. Perhaps others may like to use the same method, so I will further explain it. For purposes of comparison only, and to eliminate difficulties due to irregularities in size of wire and air spaces, I speak of '10-mesh' as cubes of 1/10-in. on a side and '100-mesh' as cubes of 1/100-in. on a side. Under these conditions, then, material of 2.5 sp. gr. would present the surface as indicated in the table above.

If, now, in any screen test the percentage remaining on each screen is multiplied by the corresponding percentage area for that size, and the percentage passing 200 mesh be considered as all remaining on 250 mesh and multiplied accordingly, this will determine the average screen size. The sum of all these products gives the total surface exposed for contact with solution. Take the following example:

On	10-mesh	1% × 3.03 =	0.0303%	of total area.
	40 "	4% × 12.12 =	0.4848%	"
	80 "	8% × 24.24 =	1.9390%	"
	100 "	10% × 30.30 =	3.0300%	"
	150 "	15% × 45.45 =	6.8170%	"
	200 "	20% × 60.60 =	12.1200%	"
	250 "	42% × 75.75 =	31.8000%	"
			56.2211%	

From this it may be concluded that the screen tests indicate material of average size a trifle coarser than 190 mesh exposes 56.22% of the maximum surface possible. Although not strictly correct, still, when dealing always with the same ore, it may be said 56% of the gold and silver content of the ore would be exposed by such size material for complete extraction.

In my Pachuca-tank test, the charge showed material equivalent to 146 mesh, assaying 237 gm. Ag per metric ton, in a dilution of 1 to 1½, yielding 77% extraction in 36 hours' agitation. This implies 44.24% (14.6 by 3.03) of the total surface was exposed or 104.84 gm. Ag per ton were readily accessible to the solution. In the mechanically-rotated paddle-agitation assisted by one 4-in. air-lift, the charge showed material equivalent to 187 mesh, assaying 397 gm. per metric ton, in a dilution of 1:35, yielding 70% extraction in 33 hours' agitation. The surface exposed was 56.66% (18.7 by 3.03) of the total possible, and 224.7 gm. Ag per ton were readily accessible for dissolution.

I claim that in the Pachuca-tank treatment 32.76% (77%—44.24) of the content was extracted from imperfectly exposed Ag; and that by the other treatment only 13.34% (70—56.66) of the content was extractd from imperfectly exposed material.

Inasmuch as the two charges did not assay the same, further investigation is desirable. In the first case, to get 77% of 237 gm., 182.49 had to be dissolved, and of this, 104.8 were free—hence, 77.7 gm. was dissolved from imperfectly exposed material assaying 132.2 gm. per ton (237—104.8), making the extraction of this material only, 58.8%. Applying the same methods to the mechanical-agitation experiment, it is found 277.9 gm. had to be dissolved out of 397 present to get 70%, of which 224.7 gm. was free, leaving —53.2 gm. (277.9—224.7) to be dissolved from a total of 172.3 gm. of imperfectly exposed Ag, which corresponds to an efficiency of 30.7% on this imperfectly exposed silver. It is obviously easier to obtain any given percentage extraction upon a high-grade ore than on a low-grade ore, so the limit to which tailing-value may be reduced in a given time is of more importance than the mere percentage extraction obtained. Thus it becomes clear that the extraction by the air-lift tanks is far more satisfactory than that made in rotary paddle agitation. The latest advance in this direction is the Just process in which the charge is practically only kept in a distended condition with just enough disturbance to cause diffusion of the enriched solution from the surfaces of the slime particles to the general mass of solution occupying the interstitial space. The advantages offered by the Just process over the Pachuca-tank treatment are many, and obvious to all engaged in operation of cyanide plants. Its universal adoption is only dependent upon further proof that the silica-sponge false-bottom will not gradually become choked. Under normal conditions, it appears to be free from this danger, as only air is passing through it and no lime-water gets to the pores. But how about the effect of frequent shut-downs for

lack of power so common in many camps? When the blower stops, the charge begins to settle and the slime and lime-bearing solution are bound to find their way into the porous tiles to a considerable degree. With the pressure of a well settled slime charge it would be strange if the pores of the tile did not become more or less clogged and the hydraulic lime, so universal in Mexico, is likely to make a pretty firm deposit in a short time. After such a shutdown, it is possible that the extra air-pressure necessary to break through the densely packed sandy slime of a re-ground product may occasionally break a tile. Or it may be that the pressure required to break up the charge will be greater than that obtainable from the blower used. This would be a great disadvantage indeed, if compressed air had to be used from time to time, as it is one of the strong points in favor of the Just process that it operates perfectly with a blower, instead of a compressor. The moderate volume of air used in ordinary practice must, no doubt, be a great factor in reducing cyanide consumption to a minimum.

Soluble Gold Slime

(September 24, 1910)

Sir—In an earlier number of the *Mining and Scientific Press,* J. D. Hubbard, in commenting on zinc-box practice at Taracol, Korea, affirms that gold in the boxes is precipitated in an allotropic form and so rendered insoluble in a cyanide solution. From the appended data which I gathered while with the King of Arizona Co., it would seem that such a state of solution should be attributed to some local occurrence rather than considered a universal condition or property of the solution. No attempt was made to obtain the exact rate at which the gold dissolved. It is to be noted, however, that nearly one-half of the total amount was dissolved and removed in the first solution charge. The slow solvent action at the close may in some measure be ascribed to occlusion of the fine gold particles in a gray flocculent residual precipitate, presumably originating from base metals in the slime, and which became lighter in color with each successive decantation of solution. In carrying out the experiments, series I, 0.20 gm. of gold slime was covered in a bottle with 300 c.c. of the solvent and occasionally agitated, then settled and at the close of the stated treatment period, decanted through four filter papers and the filtrate assayed. The heavy slime remaining practically undisturbed on the bottom was then covered with the second solution charge. The daily working solutions taken from the sump were used in the tests. While time and facilities available did not permit of exhaustive experiments, there were no exceptions to the conclusion that gold precipitate is to some considerable extent soluble in cyanide solution.

SERIES I

Chg. No.	Treat-ment. Hours.	Sol. titration. Lb. KCN, ton. Before.	After.	Sol. assays. Mg., ton Before.	Mg., chg. After.	Assay. Residue. Mg. Tot'l.
1	24	3.0	1.9	0.002	13.2	...
2	24	4.1	....	0.005	8.5	...
3	24	3.9	3.4	0.005	4.1	...
4	24	4.4	4.1	0.004	1.1	...
5	24	4.0	...	0.002	0.1	....
6	24	5.0	...	0.004	0.1	1.5

J. E. CLARK.

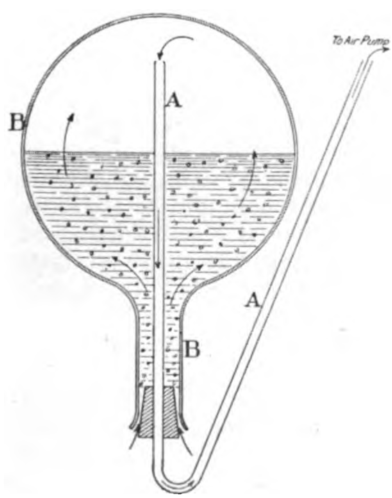
Polaris, Arizona, August 10.

LABORATORY AGITATION APPARATUS

By George A. James

(September 24, 1910)

The apparatus illustrated below is designed to furnish a simple method of duplicating the operation of the Pachuca agitator for laboratory experimental work. In the figure A is a tube bent, as there shown and of such length that it will reach quite close to the bottom of flask, B, when introduced through the loose fitting cork, and which extends about the same length outside the flask. I select for the cork one that is porous, or, if necessary, scar the cork so that air can pass between it and the neck of the flask, but do not leave sufficient space to let the contents of the flask pass out when the same is inverted. The material for treatment is introduced and the cork is entered loosely enough to permit the passage of air, but tight enough not to permit it to fall out under the weight of the contents of the flask. Tube A is then connected up with a Bunsen, or Richardson suction pump, and the flask, with contents, inverted. It will be seen that this produces a partial vacuum, which in turn draws the air around, and through the cork, and this, by being passed through the pulp, gives any degree of agitation and aeration.



Where I have a number of tests to run simultaneously, I use a tight fitting cork, and instead of the air passing around it, a bent tube which just enters above the cork, and which extends above

the level of pulp, outside of the flask is substituted. This can be connected to another apparatus, the air passing from one flask to the other in sequence.

(December 17, 1910)

The Editor:

Sir—In the *Mining and Scientific Press* of September 24, 1910, George A. James gave a method of air agitation for laboratory experiments, from which method I evolved the following scheme.

Instead of a flask turned upside down, a Squibb's separatory funnel is used. The ore and solution having been introduced into the funnel, close the funnel with the glass stopper, and mix thoroughly by shaking. The funnel is now placed in a stand, and the glass stopper replaced by a one-hole rubber stopper, which is connected by a glass tube and heavy wall rubber tubing with a filtering pump. The pump is started, thereby creating an incomplete vacuum on top of the solution in the funnel. Slightly opening the lower stopcock has now the effect of drawing the atmospheric air through the pulp and solution, resulting in efficient agitation. More than one test can be made at the same time by connecting the lower stem of the first funnel with the top of the second one, and so on, admitting air to the stopcock of the last funnel. In case of light slime, where only slight agitation is required, ordinary flasks may be used with two-hole rubber stoppers. The air is admitted through a glass tube (through the stopper), nearly reaching the bottom of the flask and is pulled away through a tube (in the other hole in stopper), which goes only a short distance in the flask and does not reach the top of the solution. The contents of several flasks can be agitated simultaneously by connecting the tube for admitting air into the first bottle, to the tube removing air from the second one, and so on.

B. W. BEGEER.

San Francisco, November 23.

MAKING LABORATORY CYANIDE TESTS

By George A. James

(December 30, 1911)

It has been my object to devise an apparatus for laboratory cyanide tests that will duplicate the conditions of actual practice, and that will handle quantities large enough to obtain accurate results, in checking solutions and tailing after treatment. I have used such an apparatus over a year with gratifying results.

The accompanying sketch shows the apparatus now in use. A large glass tube, *A*, 60 in. long and not less than 1¼ in. internal diameter (1½ in. is preferable) at the lower end, is covered with a filter medium of cloth or canvas, which is held in place by a tight-

fitting ring of metal or by rubber bands. The measuring flask, *B*, should have a neck small enough to enter the glass tube and a capacity of 2000 c.c. This is fitted with a cork, through which two small tubes enter the neck of the flask. One protrudes about an inch and the other must be at least three inches longer. The purpose of this arrangement is to provide a self-regulating feed to the percolation tube, *A*, and to prevent the contamination of the solution by the laboratory gases.

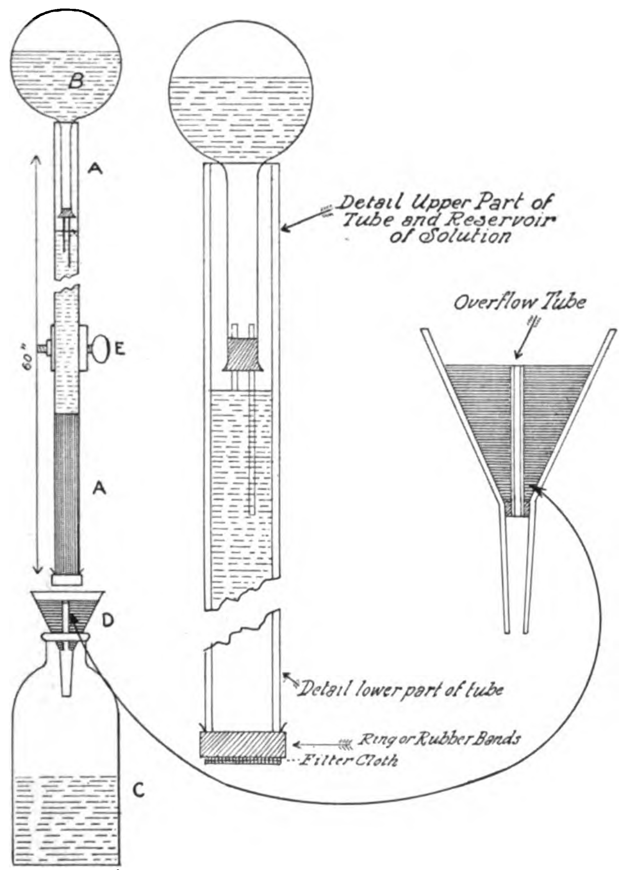
As the shorter tube provides the only inlet for air, its length regulates the height of the liquid column in *A*. Its length can be regulated as desired, but the two tubes must differ in length sufficiently to assure a flow of solution. A difference of three inches is sufficient. *D* is a funnel with an overflow tube covered with a small piece of rubber tubing, which may be pressed into the neck of the funnel, thus retaining sufficient solution for tests. This always represents the last liquid received, and can be allowed to enter the solution reservoir by releasing from the neck of the funnel, or it can be withdrawn if so desired. These large tubes are very susceptible to changes of temperature and should be protected. This can be done by pasting thick paper over them, allowing enough glass exposed for observation.

Before using the tube for experiment, it should be graduated. As the tubes are not always of regular diameter, it is well to mark them permanently at the points reached by successive additions of not more than 500 c.c. of water. This provides a simple method for determining the specific gravity of ore charges, when using the tubes for experiments. In using the tubes the ore charge is introduced, preferably with a funnel, from the top. The feed regulator, B, is then inverted in the neck of the tube, and no further attention is required. It is well to bend the point of the discharge tube of the supply flask so that the liquid will not drop directly on top of the ore charge, but against the wall of the tube *A*.

After introducing the ore charge and solution as described, the time is noted. For this purpose an experiment card is provided for each apparatus, which it is best to hang conveniently at hand. This card should be provided with columns in which are entered all the amounts of liquid removed for tests, the total content of solution in chemicals, and the total content of the ore charge in gold and silver. I use 1 kg. of ore in most tests, and estimate the total milligrams of gold and silver contained. As soon as the solution is exhausted from the solution-reservoir, the speed of percolation can be noted. This can be taken as correct for similar depth of ore-beds, no matter what diameter the beds may have, and can be safely rendered as the speed of percolation for similar charges on a large scale. It is important to keep track of this detail during the time of the experiment. When the entire solution has passed through the ore charge, the amount recovered is measured (with allowance made for solution appropriated for tests), and the difference from the original amount charged to what is held by the ore. If wash water has been used, the amount held by the ore charge

must be known, and its effect considered in increasing the quantity of solution, with its bearing on the cyanide strength in tests, also in precipitation for gold and silver extraction.

It is my custom to precipitate the gold and silver after the passage of each total quantity used, with a weighed amount of zinc dust, remove the precipitate by filtration, and reduce in the assay furnace, thus obtaining the actual metal leached, and introducing the effect of zinc found in solutions used in practice. These incom-



LABORATORY CYANIDE APPARATUS

plete recoveries are compared to the original amount in the ore charge, their proportion to the remaining metal in the tube noted, and furnish an accurate comparison of the progress of the experiment. After a few tests a factor is found that will make them accurate enough to determine the progress of the work and indicate the point desired. The assays of the tailing remaining in the tube should be expected to check closely.

Where it is desired, a portion of ore charge can be removed by a sampling device. I use for this purpose a glass tube, attached to a rubber tube sufficiently long to permit its entrance from the top of the percolation tube. This is allowed to rest on top of the ore charge. By suction it will enter into and draw up the pulp. This can be withdrawn, with its contents, by pressing the rubber tube to prevent the entrance of air. By slipping a heavy rubber band over the filter-cloth of the percolation tube, to form an air-tight connection, the end may be introduced into a suction flask and the rate of filtration accelerated by an air-pump.

(February 17, 1912)

The Editor:

Sir—The description of apparatus for use in making cyanide tests, published in your issue of December 30, calls for comment in spite of the fact that the author, George A. James, states that the duplication of the conditions obtaining in actual practice was aimed at.

In the leaching of gold-silver and, more especially, silver-gold ores the ratio of depth of sand to satisfactory extraction is an important question. The dimensions of Mr. James' apparatus limit the practical application from any results obtained to a depth of sand of about 3 ft. To carry out experimental work on 3 ft., or less, and to expect similar extractions with 10, 15, or even 20 ft., is a common mistake. In many instances the residual content varies directly with the depth leached, and exceptional cases have been known where a residue sample taken from the bottom of a leaching vat, after prolonged treatment, has been found to contain a higher metal content than did the original charge. My own opinion is that the great majority of estimations made to determine possible extraction by leaching are of little value on account of the absurdly small amounts of sand taken for treatment in the preliminary experimental work, and the production of data obtained from investigations on a depth of material too shallow to demonstrate the chemical changes in the solution, and resultant effects on extraction, occurring in an ordinary leaching vat.

The second point needing comment is that Mr. James' arrangement, in avoiding "the contamination of the solution by the laboratory gases," prevents its frequent aeration by natural means. Where reducing action is evident, an advantage is gained by the periodic draining of the charge, and the intermittent application of solution washes.

The third point to which I would draw attention is that Mr. James' apparatus possesses no contrivance by which the affluent can be shut off, or by which the rate of leaching can be controlled. The preliminary 'soaking' of the ore in strong solution necessitates some valve arrangement, and if Mr. James' method of uncontrolled gravity leaching were adopted in practice with a coarse clean sand the pumping expense would be prohibitive. The determination of

maximum percolation rate is of secondary importance to the determination of *efficient* percolation rate, since the rushing of solution through a bed of sand results in the handling of, and precipitation from, large quantities of low-grade solution, the production of base precipitate and its attendant expense and trouble.

I am indebted to my early laboratory training in such work and I have never departed from the use of ordinary 4-in. piping in the experimental determination of cyanide extraction by leaching. The pipes are fitted with bored flanges, at one end for suspension and at the other for the filter bottom. The latter is kept in place by a third flange with insertion joint, and fitted with a plug tapped for a small leaching-control pipe, an arrangement incidentally providing for the upward charging of strong solution. The pipes should be cut or compounded of a length equal to the height of the vat it is proposed to use in practice; or, if the dimensions of the latter have not been decided upon, experiments should be made with various lengths. In all cases, and at the conclusion of treatment, bored samples should be taken at the top, and also at the bottom of the pipe, and the assay results compared. Equality of residual content would indicate the possibility of treating larger bulks of ore by an increase in the height of percolation. A higher residue at the bottom than at the top would indicate the necessity of reducing the leaching depth, or of resorting to double treatment. In most cases the latter method is preferable to single treatment. Head and tail samples should be taken after the charge has been thoroughly mixed, before and after treatment.

While regretting the destructive nature of the criticism on the article, I would like to add my agreement in the matter mentioned by Mr. James as to the importance of the re-use of barren, and not freshly prepared solution, in experimental work; and I take this opportunity of adding that cyanide consumption estimations should not be calculated experimentally in the manner advocated in treatises on cyaniding, but should be based on methods involving the titration for total cyanide percentage in the solution before treatment, and after treatment *and precipitation of metals;* the experiment should be made, not with distilled water and pure potassium cyanide, but with barren solution containing the normal amount of the double cyanide of zinc and potassium, and any other salts induced by the continued use of the solution in the treatment of similar ore; and the actual amount of commercial cyanide, of analyzed KCN content, needed to re-standardize the solution should serve as a check on the titration difference.

London, January 15.

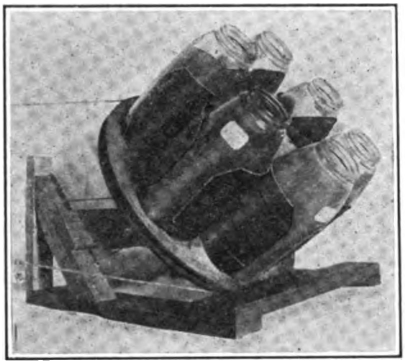
A. W. ALLEN.

(March 23, 1912)

The Editor:

Sir—The question of long-pipe percolators in making cyanide extraction tests, as discussed by George A. James in your issue of December 30, 1911, and by A. W. Allen, February 17, 1912, is in-

teresting. As more and more rebellious ores are being treated by the cyanide process, the question of laboratory tests and the appliances for making them becomes of increasing importance. The tall iron-pipe percolator has been used in this country, but I have never seen it so conveniently arranged as Mr. Allen suggests, with the easily separated flange couplings. It seems to me that it is not alone the determination of the leaching rate that makes the pipe percolator useful. For instance, if the leaching rate is so rapid that an unnecessarily large amount of solution would have to be passed to the zinc-boxes, the leaching rate can be cut down at the valve and the effect on the solution noted. An examination of the affluent solution from a deep charge of ore often reveals important facts. If the affluent solution is in proper condition, little uneasiness need be felt as to the extraction in the lower portion of the vat. I have always considered it quite important to keep a percolation test leaching continuously where there is the least danger of the solution becoming too low in protective alkali or picking up alkaline sulphides to a considerable extent during its passage through the charge. The test can conveniently be kept leaching through the entire night by filling a can with solution and inverting it over the percolator after the manner of the flask suggested by Mr. James. However, the greater part of the work can usually be done by agitation tests, and on account of the large number that may be run simultaneously with little outlay for equipment, I think they are to be preferred, using the percolator tests as a final check rather than the principal method. Of course, everybody makes 'bottle tests,' but they are too often regarded as a rather unimportant preliminary. In making the so-called bottle tests, too small a quantity of material is usually taken.



CONVENIENT AGITATION APPARATUS

The agitation apparatus I prefer to use consists of a wooden disc rotating at a speed of not more than 5 to 8 r.p.m., the shaft being set at such an angle that the axis of rotation will coincide with the hypothenuse of a right triangle whose vertical altitude is 1 and its base 2. The top of the disc is provided with sheet-steel clips to removably hold six or more two-quart (half-gallon) Mason fruit jars, the bottoms of the jars resting flat upon the disc. Enough solution can be used in this apparatus to make zinc-dust precipitation practicable, and in case separate treatment of sand and slime is contemplated they need not be separated in the earlier stages of the work. The tests can be put on the agitator near the close of the day, run during the night, and will generally be ready for assay

in the morning, thus making it possible to turn off a considerable amount of work in a short time and with little labor. By means of this apparatus I once determined with a fair degree of accuracy what might be expected to happen in the bottom of a deep, slowly leaching charge, by filling the jar full, sealing it with the cap, and titrating the solution at intervals. Of course, in making a test of this kind, due account must be taken of the relatively large amount of solution in the agitator. For instance, if the ratio of solution to ore is 5 to 1, and the voids between the grains of sand in the leaching vat would allow the vat charge to consist of 50% solution (a ratio of 1 to 1), then whatever changes are found in the solution at the end of the test should be multiplied by 5. I do not know how I could have determined this point by means of the usual quart percolators used in this class of work.

It is perhaps needless to add that in making tests upon coarsely crushed material, either the half-gallon jars or ordinary glass percolators are liable to lead one considerably astray on account of the poor sampling which inevitably results. A convenient and easily remembered rule for the size of test samples is to square the diameter of the largest grains, measured in sixteenths of an inch. The result will be the number of pounds of ordinary mill rock that should be taken to accord with good sampling practice. For instance, if the stuff is crushed to a five-sixteenths-in. screen opening, 25 lb. would be sufficient, but if the screen opening is seven-sixteenths, the amount required would be, in round numbers, 50 pounds.

Boulder, Colorado, March 8. JOHN RANDALL.

LABORATORY CLASSIFIERS

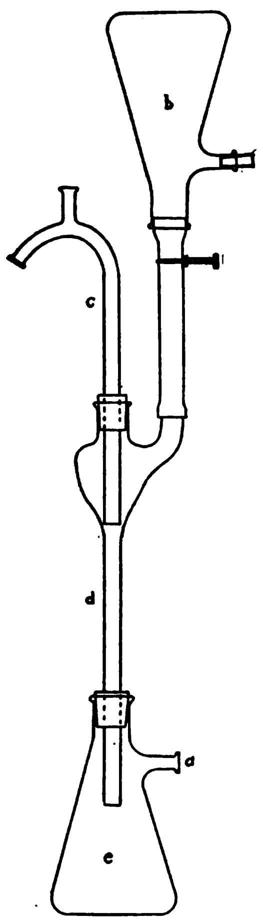
(May 11, 1912)

The Editor:

Sir—Elsewhere in this issue* W. J. Sharwood describes a simple and cheap type of laboratory classifier, which he employed in the classification of zinc-dust. The apparatus there described has the great advantage of being constructed from material available in every laboratory, and served admirably the purpose for which Mr. Sharwood intended it, but if others were to employ it for general work faulty results would be obtained that might vitiate the conclusions reached. For classification of ores it is evident that the velocity of flow to be considered is that at the lip where the material is overflowing, rather than in the body of the tube, and it is likewise obvious that the amount of ore that can thus be classified is too small for convenience in experimental work, since if the ore present in the tube exceeds one-tenth of the volume of the ascending current of water it will give rise to interstitial currents which interfere with separation.

Since the present trend of ore-dressing practice is toward closer classification, and closer control of mill operation in the light of laboratory experiment, it may be worth while to call atten-

*Reprinted in this volume under 'Precipitation and Clean-Up.'



MUNROE LABORATORY CLASSIFIER

tion to the convenient laboratory classifier shown by the accompanying illustration, which was devised by H. S. Munroe, professor of mining in Columbia University, and first described in the *School of Mines Quarterly* April, 1901, page 303. It consists essentially of two Erlenmeyer flasks of convenient size, a specially constructed feed tube, *d,* and a classification tube, *e.* These are connected as shown. In use, *a* is connected with a source of water supply of constant head, and by measuring the outflow from *c* for a given period and dividing by the area of *d,* previously determined, the rate of flow is computed. The ore, mixed with water, is placed in *b* and the pinchcock opened sufficiently to allow the desired amount of feed, which should not exceed one-tenth the volume of the rising current in *d.* As the ore passes down the feed tube, water passes up into *b* to take its place, thus tending to prevent choking. A small quantity of air at the top of *b* aids in this, as by gently squeezing the rubber tube a pumping action is produced which will free the tube even when choked with quite coarse sand. By re-treating the sized product with successive currents of increasing velocity any number of classifications desired can be made. Both in ordinary ore-dressing and in cyanidation the screening tests which are commonly employed are not so significant as a determination of classification ratios would be. Furthermore, with screens it is impracticable to obtain a closer separation than, for example, through 150 mesh, on 200 mesh, through 200 mesh, while with a classifier any desired degree of accuracy can be obtained.

THOMAS T. READ.

San Francisco, May 6.

SPECIAL PROBLEMS

A CYANIDING PROBLEM

(August 13, 1910)

The Editor:

Sir—There are many contributions to the technical press dealing with the cyaniding of gold and silver ores, and numerous books on the subject, all of which give valuable data, metallurgical, mechanical, and economic, but there are still phases of the treatment of certain refractory ores which none of these authors mention, or, if they do, in only a casual manner, throwing little light upon vexed questions. I have in mind an ore consisting of a quartz gangue in which occurs pyrite in abundance, with a relatively small amount of galena, blende, and chalcopyrite. The value is chiefly in gold, which to some extent is free, that is, it may be panned, but accompanying it is silver, which mostly occurs as a complex antimonial compound. This ore has thus far successfully resisted all efforts to treat it either by amalgamation or cyanidation, the tailing still carrying nearly half of the original value of the ore. If some one of the numerous contributors to the extensive literature of the cyanide process would tell how the antimony in this ore may be eliminated, or at least rendered innocuous, it would be appreciated by many having mines producing ores of similar character. It is quite possible that roasting before cyanidation, or roasting in connection with the chlorination process, would solve the problem, but the additional cost of handling and roasting in a region where fuel of any kind is far from abundant makes the treatment of the raw ore desirable, if not necessary, for reasons of economy. Then, too, it is not always an easy matter to secure the services of an expert roaster, such as would be imperative in the handling of an antimonial ore. Arsenic is bad enough, but antimony is worse.

MINE OWNER.

Tuscon, Arizona, August 5.

(August 27, 1910)

The Editor:

Sir—In your issue of August 13, I note a communication signed 'Mine Owner,' Tucson, Arizona, in which the writer, after stating that among the contributions to the technology of cyanide treatment, little or nothing is to be found concerning the economic treatment of antimonial ores says, that "if some one of the numerous contributors * * * * would tell how the antimony in this ore may be eliminated, or at least rendered innocuous, it would be appreciated by many having mines producing ores of similar character." As a member of that body of men who have put in years of study, research, and hard work on the treatment of gold and silver ores, I should say that the mine owners certainly would appreciate such information, given them gratis, but would like to ask 'Mine Owner' whether he considers it good business for metallurgists to give away the results of their work and experience in this manner? While it is true that the surgeon or physician gives any discovery

which he may make, to the profession, through the medium of the technical press, the analogy ends there, for the profession is protected by law and the methods so given out cannot be taken up and practiced by pharmacists and prescription clerks the world over. Let us take the case of manufacturing companies who employ technical men in their experimental departments. I have myself worked in this capacity and I know just how zealously all so-called 'trade secrets' are guarded. If this be good business in the case of the company, why is it not the same in the case of the individual, who makes his living by his knowledge of such matters? Information of this character is valuable, and although the methods of extraction available for the class of ores cited are rather involved and more costly than simple treatment, satisfactory results can often be obtained by the man who knows how, where failures have preceded, and the extraction is almost entirely a business question, which a careful and competent metallurgical examination will answer in the surest possible manner.

METALLURGICAL ENGINEER.

La Jolla, California, August 16.

[We happen to know that our friend who signed his name 'Mine Owner' has strong conviction on the point raised by 'Metalurgical Engineer,' and for the present we shall leave the answer to him.—Editor.]

(September 3, 1910)

The Editor:

Sir—I will not pretend to be astonished at the caustic reply of 'Metallurgical Engineer' to my inquiry for information in regard to the treatment of auriferous and argentiferous ores carrying antimony, for I am not. I have had the opportunity to read most of the works on the cyanide process, including those by Scheidell, Bosqui, James, Louis, Park, Clennell, Julian and Smart, and several others, besides numerous classified contributions on the subject from every gold mining field of importance in the world, and must confess that the most dilligent search has failed to discover anything of consequence, and I may add, of value, on the question in point. Since this is so, our friend, 'Metallurgical Engineer,' is to be congratulated on the possession of the secret—or are all of these excellent and experienced engineers holding this particular card up their several sleeves, with a view to turning it to good commercial advantage? This I do not believe, for almost every phase, it would seem, of the application of the cyanide process in its many modifications to ore treatment has been elaborated and all secrecy banished, in the lengthy, and sometimes acrimonious, discussion of various matters. It would be strange indeed if this one lone vexed problem were the only one left undebated. I learned many years ago that an attempt to create a 'corner on knowledge' was almost hopeless, and I think this instance will prove to be no exception.

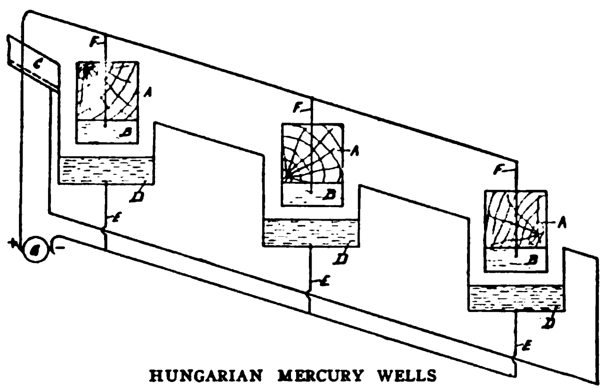
MINE OWNER.

Tuscon, Arizona, August 29.

(September 10, 1910)

The Editor:

Sir—In your issue of August 13 appears a communication signed 'Mine Owner,' from Tuscon, Arizona, in which reference is made to quartz gangue, abounding in pyrite, with small amounts of galena, blende, and chalcopyrite, the value being chiefly in gold, which may be panned; the further statement follows that after amalgamation and cyanidation the tailing was of about one-half the value of the head; your correspondent states that silver is also present in a complex antimonial compound, and attributes the low extraction to the presence of antimony, and desires to know if there is any remedy besides roasting.



HUNGARIAN MERCURY WELLS

Three electrolytic sodium-mercury cells, forming a portion of electrolytic amalgamating sluice; longitudinal sectional view. A, baffles for deflecting gangue on mercury surface; B, graphite electrode (anode); F, positive connection with generator, G; D, mercury connected as cathode with negative lead, E; C, launder delivering pulp, water, and salt solution.

I would suggest that a remedy may be found in the use of the electrolytic sodium-mercury cell for amalgamation in which a series of mercury cells or riffles (Hungarian mercury wells) are arranged in the form of an electrolytic amalgamating sluice, the mercury being excited by a low voltage current (five to ten volts) of high amperage (twenty to thirty amperes per square foot of mercury surface). When such a device is adjusted to the existing conditions, it has been found that all gold and silver in a metallic state not encased readily amalgamates. Pulp and slime are simply passed over the device and the metals are automatically extracted in the form of amalgam. The construction of the apparatus prevents any blanketing of the mercury and fouling or flouring is impossible. To obtain these results in the electrolytic mercury cell a definite amount of salt solution is fed into the water passing over the sluice; sodium amalgam is formed and any required strength is constantly and automatically maintained. The construction of the mercury sluice is illustrated in the accompanying design, which represents a longitudinal section of three cells. In such a device, properly constructed

and adjusted, all free gold and silver, regardless of the presence of sulphur, arsenic, copper, antimony, or other refractory elements, will readily amalgamate while the gangue is passing over three or four lineal feet of excited mercury surface; such a device has a capacity three times as great as a mill plate of the same width.

To determine the amount of free metal in ore at any given mesh, assay a sample; then treat a similar sample with strong sodium amalgam. The sodium amalgam should be strong enough to amalgamate copper wire instantly. After agitating for twenty minutes carefully draw off the water, pulp, and slime, evaporate and again assay; the difference between these assays shows the free gold present, which may be extracted by a series of electrolytic cells. As only a low voltage current is needed for the electric amalgamator, the cost of operating such a device is trifling; a five horse-power generator will furnish current to treat the product of a 40-stamp mill. I might add that these statements are based on the results of scores of demonstrations, and there is ample evidence that the electrolytic sodium-mercury cell will extract all released gold regardless of the physical or chemical conditions of the gold or associated elements. The basic patents on electrolytic amalgamation have all expired.

ELMER ELLSWORTH CAREY.

San Jose, California, August 16.

(September 17, 1910)

The Editor:

Sir—In reply to 'Mine Owner's' inquiry for a process for the treatment of auriferous and argentiferous ores containing antimony, it appears to me that the difficulty should not be insurmountable, always provided, of course, that the ore will stand the treatment charges. J. S. McArthur proposes a pretty process, consisting of dissolving the antimony—that is, when the antimony occurs as sulphide—by first leaching the ore with a solution of caustic potash, precipitating the antimony from the solution as sulphide by passing carbonic acid gas, and then revivifying the solution, ready for use again, with caustic lime. If there is no other deleterious content, the ore—now free from antimony—may be treated with cyanide in the usual way. This process is not applicable to ores in which the antimony occurs as metal. Antimony may be removed from an ore by roasting if a reducing agent such as coke dust be added from time to time during the process. During the roast, antimony sulphide or metal is converted into Sb_2O_3 and Sb_2O_5, which are volatile, and Sb_2O_4, which is fixed; the function of the reducing agent is to reduce any Sb_2O_4 that may be formed and would otherwise remain with the ore, to Sb_2O_3, which, being volatile, will pass away up the flues. An article of mine, that appeared in *Mining and Scientific Press* April 28, 1906, and was reproduced in T. A. Rickard's 'After Earthquake and Fire,' gave the results of a number of experiments which may interest 'Mine Owner.'

F. H .MASON.

San Diego, California, September 7.

(September 24, 1910)

The Editor:

Sir—I note with interest the letter of 'Metallurgical Engineer' in your issue of August 27, commenting on the request of 'Mine Owner' for a method of dealing with antimonial ores by means of the cyanide process. 'Metallurgical Engineer' believes that it would be foolish for one in possession of the desired information to make it public gratis, as such knowledge is the capital of him who has it. The standpoint is natural, and it is easy to understand its acceptance by the great majority of people. This is doubtless the reason why the columns of the technical press rarely give notice of the details of a really important new idea, and it is also the reason why the records of the patent office are so clogged with applications for patents on inventions that are considered important by their fond originators. The idea, however, seems to me rather near sighted. It is true that the mine owner may be required to pay a high price for his information originally, but if mine owner's neighbor, having the same kind of ore to treat, requires that information, the chances are that he will get it by watching operations. Valuable information of that sort is bound to spread in one way or another, and it is futile to try to prevent it. On the other hand, if this information were published freely it is more than likely that it would require a trained metallurgist to apply it, and the man with the knowledge would get his job after all. There need be little fear that such information would be appropriated and used by novices. It requires a scientific mind and scientific training to apply scientific truths, and while the metallurgist is not protected by statute, he is amply guarded by the law of human capability. No one who has read the pleas of T. A. Rickard for the free publication of technical information can pass the matter by without thought. And the more thought given to the subject, the more clear it will appear that the one who gives his information is little likely to suffer by it. There is, however, a more important reason why 'Mine Owner' cannot receive an answer to his question in the columns of the press, and that is that his information is too meagre. Some ores containing antimony are easily treated with cyanide and others not so easily. It depends not only on the amount of antimony, but its condition and combination with other elements, so much so that it would require a whole book to include all the possible occurrences of the element in ores.

H. A. MEGRAW.

San Luis de la Paz, Gto., Mexico, September 2.

(October 22, 1910)

The Editor:

Sir—Replying to 'Cyanide Problem' in the issue of August 13, it may be assumed that the premise in this case is "an ore, consisting of a quartz gangue, in which occurs pyrite in abundance, with a relatively small amount of galena, blende, chalcopyrite," gold, and a complex silver-antimony compound; and that the desideratum is,

to extract that amount of the several metals or minerals, contained in this ore which will show the greatest profit over the cost of operations. The primary conditions to be confronted are that direct amalgamation of the ore, and doubtless, amalgamation at any time, is prohibited by the presence of galena, blende, and the complex silver-antimony compound in the ore; that direct cyanidation of the ore is impracticable, due to the presence of galena, blende, complex silver-antimony compound, and other cyanicides; that any roasting treatment of the ore is prohibitively high, due to cost of fuel and other considerations. Under the conditions imposed we are confined to the process of cyaniding the ore, and as a secondary condition to the use of this method of treatment, we must 'eliminate' the cyanicides, antimony, galena, blende, etc., or at least render them 'innocuous.' To 'eliminate' the elements in the ore detrimental to cyanidation, naturally implies their removal by some mechanical means; and to render them 'innocuous' implies a change from their present state of association to a state not detrimental to the action of the cyanide in dissolving the gold and silver contained in the ore, namely, a chemical change. Now we must decide upon which method or combination of methods should be followed to 'extract that amount of the several metals or minerals contained in the ore, which will show the greatest profit over cost of operations.' As agents for eliminating from the ore, the antimony compound, galena, blende, etc., we may employ classifiers or concentrators. As an agent for rendering the antimony 'innocuous' we may add caustic alkali to the ore; or, we may crush the ore to such a size that a minimum amount of antimony and a maximum amount of gold and silver will be exposed to the action of the cyanide. In regard to the latter proposal I have read that, in Borneo, under the direction of Alfred James, good extraction of gold was obtained from an ore containing considerable antimony and arsenic, by breaking the ore to a size, varying between that of a lemon and a walnut, and treating with a very dilute solution of cyanide. In regard to the second proposal, the alkali, in combining with the antimony, also combines with the oxygen of the air; thus if the alkali-antimony compound be formed in the cyanide solution, the potency of the solution for dissolving would be diminished by the paucity of uncombined oxygen in the solution, as present in air. In regard to the first proposal, there does not appear to be any reason whatsoever that the pyrite, chalcopyrite, blende, galena, antimony-silver compound, and gold should not be removed from the gangue of the ore by concentrating tables, to almost any degree of refinement desired. Now, neglecting for purposes of discussion the last proposal, we must determine the comparative economies between crushing the ore to concentrate, followed by cyaniding the concentrate, and crushing the ore to cyanide, the addition of the alkali preceding cyanidation, or between either of these methods and a combination of them both. Naturally it is not possible here to point out the limit to which any one step in a series of steps composing a method of extraction should be carried out or what combination of steps should form a method. But,

provided the third proposal be not applicable, from what data we have at hand, it would seem that crushing, followed by concentration, followed by treatment of the tailing with alkali, by agitating the alkali with the tailing with compressed air, followed by cyanidation, would give good results. As must be evident, the above is but a 'speculation' upon the problem given, and is, in fact, an expression of the method of solution proposed for a similar problem, once expected to be confronted, but which finally was not encountered.

LEE FRASER.

Punta Arenas, Costa Rica, September 21.

(November 26, 1910)

The Editor:

Sir—A statement of the principles embodied in Designolli's process for treating gold ores containing antimony may assist in suggesting a solution to the 'Cyanide Problem.' Briefly, the process consists in crushing ore, or treating the ore with a solution of bichloride of mercury and sodium chloride, and for its basis rests upon the facts that when an acid solution of any salt of mercury is exposed to the action of an electric current the salt will be decomposed and metallic mercury deposited on the cathode, and when the cathode is gold, will result in amalgamation. When clean iron surfaces are in contact with a salt of mercury, in a weak acid solution, on touching them with gold the salt of mercury is decomposed, the metallic mercury thereupon amalgamating with the gold. And, that chlorine has a great affinity for antimony.

In operation, the antimony compound is decomposed, the antimony uniting with the chlorine, while the gold combines with the metallic mercury.

LEE FRASER.

Punta Arenas, Costa Rica, October 15.

Note: Commercial bichloride of mercury, or corrosive sublimate, costs 90c. to $1 per lb., and sodium chloride, or commercially, common salt, costs 2 to 4c. per pound.

A TREATMENT PROBLEM

(November 16, 1912)

The Editor:

Sir—I wish to bring to your notice the following: I have recently taken up a number of heaps of silver tailing (600,000 tons in all), scattered over the province of Junin, Peru. These are the results of the old *patio* process worked by the Spaniards on Cerre de Pasco and other silver ores for the last three hundred years. They consist of very fine material (80% passing through a 200 mesh), silver content 8 oz., SiO_2 80%, Al_2O_3 1.7%, FeO 7.7%, and a fairly

large amount of mercury. In order to ascertain the true value of the tailing, I want to obtain information in regard to the cost of extracting the silver. I should feel very much obliged, therefore, if some of your readers would inform me of the results obtained by the companies in Mexico or elsewhere that are working similar material.

HENRY VOGEL.

Cerro de Pasco, Peru, October 12.

(December 14, 1912)

The Editor:

Sir—In your issue of November 16, Henry Vogel makes inquiry regarding cost of treatment of accumulated silver-bearing tailing derived from patio or other processes, and carrying about 8 oz. silver per ton. He refers to dumps of such material, aggregating 600,000 tons, scattered over the province of Junin, Peru. Presumptbly, the physical condition of the tailing as to cleanness, location, size of individual dumps, accessibility, questions of water and power will have important bearing on costs, aside from metallurgical considerations. The conditions assigned by Mr. Vogel, namely, 80% passing through a 200 mesh, silver content 8 oz., SiO_2 80%, Al_2O_3 1.7%, FeO 7.7%, and a fairly large amount of mercury present, no not, therefore, present the problem fully enough to prevent a comprehensive and categorical reply.

I spent about three years in the testing and treatment of a large old dump of silver-bearing tailing at Cortez, Nevada, containing 100,000 tons of material and carrying about 8 oz. or silver with a little gold. The ore had previously been roasted and treated by thiosulphate. Owing to faulty lixiviation washing methods, about half of the metal content of the tailing was readily soluble in weak cyanide solution, enough more to make up an 80% recovery, which seemed to be the maximum possible, was extremely difficult to get into solution with a reasonable cyanide loss. Finally, no attempt for a high recovery was made and attention was concentrated on operating methods, to reduce the cost to the minimum. Owing to the cost and uncertainty of labor in districts away from the railroad, and numerous other resulting factors, our costs were excessive, but inasmuch as these still permitted a profit, operations were fairly satisfactory. Much preliminary testing was done by the California Ore Testing Co., and the results obtained in treatment of about 30,000 tons were closely anticipated by these tests. The general scheme of treatment was by rapid leaching, using a large total volume, as much as 5 to 1 of solution, of low strength, 0.2 to 1 lb. per ton. Excessive alkalinity aided solution and precipitation results, and about 15 lb. CaO per ton tailing was added. The extraction ranged from 45% to 55%, averaging 50% and the costs on about 15,000 tons treated, at the rate of 80 tons per day, were very closely as follows:

	Per ton.
Labor	$0.34
Solution men	0.14
Superintendence	0.08
Cyanide, ½ lb. at 22c	0.22
Zinc shaving, 0.4 lb. at 15c	0.06
CaO, 15 lb. at 1c	0.15
Gasoline power	0.06
Marketing bullion	0.05
Depreciation and amortization	0.20
Supplies and petty cash	0.05
	$1.35

Many physical difficulties, owing to adverse operating conditions, were encountered in the treatment of the tailing. It was found most essential to change the stock solution completely every month or two, owing to fouling by soluble impurities in the tailing. On using fresh solution, good results were always obtained with little trouble, but after use for several weeks the fouled solution gave much trouble, in precipitating principally. This was the only important metallurgical difficulty.

A. W. GEIGER.

San Francisco. December 3.

CRUSHING

PRELIMINARY HANDLING OF ORE AT EL TIGRE

By J. W. Malcolmson and L. R. Budrow

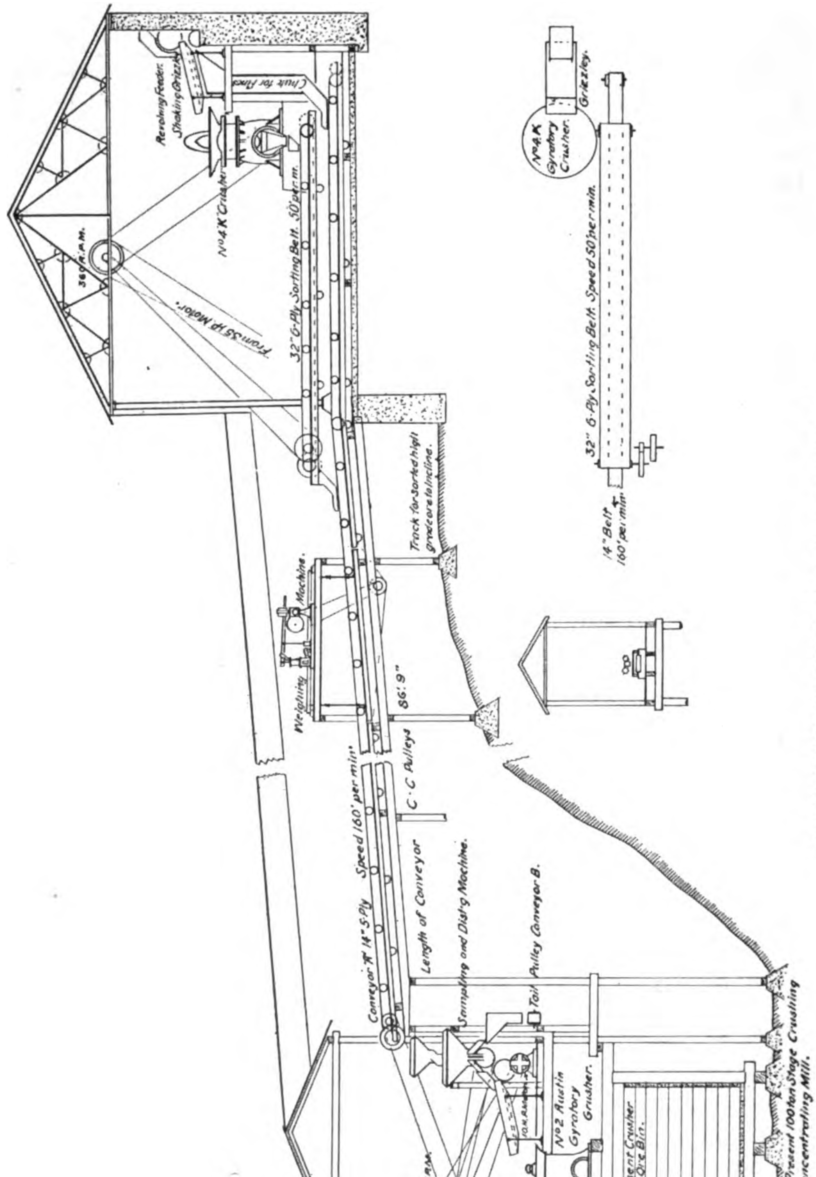
(March 16, 1912)

The El Tigre mill, belonging to the Lucky Tiger-Combination G. M. Co., is situated in Sonora, Mexico, 30 miles from any railrôad. Power is supplied from Douglas, Arizona, over a line built during the revolutionary troubles of last year, and it is interesting to note that despite the fighting at Agua Prieta, through which the line passes, not a day was lost during the period of construction. The mill, of which a general view is shown opposite, and which was designed and built by D. L. H. Forbes, was put in operation July 1, 1911.

The sorting arrangement here described was designed for a capacity of 600 tons in 24 hr., but at present is used on the day shift only. All second-class or mill ore is trammed from chutes on the lowest adit-level over 3000 ft. to a circular steel bin, of 15-ft. diameter and 30-ft. height, and having 250 tons capacity. A track carried on trestle leaves the main line, passes over the top of the bin, and joins the main track beyond. The loaded cars run over the top of the bin and, without changing direction, are carried forward to the main track, returning on the main line along the hillside to the mine. All underground tramming from the chutes to the bin is done with mules, and the present output of 200 tons per day can easily be handled with three mules during the day's shift. The cost of tramming is 11c. per ton. By changing to electric haulage, the cost would be reduced to 4c. per ton.

The circular steel ore bin has a flat concrete bottom, 8 in. thick. The bin is made in 5 sections, the lower 2 being of ⅜-in. steel; the next 2 of five-sixteenths steel; and the upper section, ¼-in. steel. The top and bottom are stiffened with rings of 1½-in. angle-iron. The ore is drawn from the bin by means of a revolving drum of 24-in. diameter and 27-in. width. The mechanism for turning the drum consists of an eccentric with a 1-in. throw operating a friction wheel of 12-in. diameter, keyed to the drum shaft. A rack-and-pinion gate above the drum regulates the amount of ore fed at each forward movement of the drum and facilitates the passage of large pieces of ore.

From the drum, the ore falls upon a shaking grizzly, 6 ft. long by 4 ft. wide, with bars, 2 by ¾ in., spaced 2 in. apart. The grizzly is operated by a quick-return eccentric drive. The undersize from the screen falls into a short chute of funnel which delivers it upon a ¼-in. conveyor-belt, traveling at a speed of 180 ft. per minute. The oversize falls into a Gates type K No. 4 gyrator crusher, which reduces the ore to a size that will pass a 2-in. ring. The crusher feeds on to a 32-in. picking-belt traveling 50 ft. per minute. This travels continuously at 50 ft. per minute, and 4 men are employed, 2 of them being ore-sorters, who stand on opposite sides of the belt and thrown down and to one side. W. W. Mein has suggested as a preferable arrangement and an intermittent movement of the belt in



ARRANGEMENT OF CRUSHERS AND SORTING BELT

order that the sorters may work to better advantage. This is used in South Africa, the belt moving several feet, then stopping for a few seconds and starting again. Such an arrangement permits each ore-picker to handle the same area under equal conditions.

The high grade ore sorted from the belt during December 1911 amounted to 6½ tons, assay value $310 per ton; the waste sorted out was 34½ tons, assay value $1.90 per ton. The ore milled, after deducting 125 tons of high grade sorted while mining, and the sorted products mentioned, assayed $20.25 per ton. The Gates crusher and picking-belt are housed in a steel-frame building. The arrangement is shown in the accompanying figure.

The ore from the picking-belt falls on the 14-in. conveyor-belt running underneath. It here joins the undersize from the grizzly and is weighed by a Blake-Dennison automatic weigher while in motion. This machine works admirably, recording the weight of the ore with an error of less than 1 per cent.

The ore from the 14-in. conveyor falls into a revolving cone distributor, which in trun discharges on a stationary cone divided into three segments. This arrangement divides the feed into three portions. One portion, representing 5%, goes to the sampling plant, and the remaining 95% falls into two compartments with an adjustable partition permitting the distribution of ore to the old stage-concentrator or to the new stamp-mill in the proportions desired.

The ore for the old concentrator falls on a second shaking grizzly with bars spaced ¾ in. apart. The undersize falls directly into the old 125-ton ore-bin, and the oversize runs through a No. 2 Austin gyratory crusher and is reduced to pass a ¾-in. ring, falling into the 125-ton bin. The other compartment of the stationary cone discharges on a 12-in. conveyor-belt traveling 200 ft. per min., and delivers the ore by means of a Robbins automatic tripper into a 350-ton wooden ore-bin above the new 20-stamp mill.

The 5% sample falls into a 7 by 10 Blake crusher and is reduced to ½-in. size. The crusher discharges into a Snyder disc sampler, eliminating 95% of the sample; the remaining 5% is passed through a 2 by 6 Sturtevant roll jaw crusher and quartered down on a steel plate until the quantity is reduced to 50 lb., when it is sent to the assay office.

GYRATORY v. JAW CRUSHERS

(February 3, 1912)

The Editor:

Sir—Can you refer me to any reliable comparisons between the two types of ore or rock crushers, gyratory and jaw? I cannot find anything comprehensive along this line.

ENGINEER.

San Francisco, January 25.

[The best data on the two types of crusher available may be found in Richards 'Ore Dressing,' Volume I. There is room, however, for much additional data. It is difficult to get a direct comparison, for the reason that both types of crushers are not generally

used in the same mill and under the same conditions. The best chance for comparison is where in rebuilding, one type has been replaced by the other. Summarizing the experience of three engineers, each having used both styles of crusher, it may be said that for large capacities the gyratory has the advantage of a lower capital cost for actual tonnage handled; ordinary renewals are not greatly dissimilar; extraordinary repairs on the gyratory may offset largely, if not entirely, its other advantages. The trouble arises from broken spindles. It is difficult to secure steel shafting of the quality needed for this use, and breakage is markedly irregular. A given spindle may last a long time, only to be replaced, when finally broken, by a rapid succession of imperfect ones. On this account, costs averaged over a short period only may well be especially deceptive. It is none the less significant that practically all of the large new plants include gyratory crushers.—EDITOR.]

ROCK-CRUSHERS AT KALGOORLIE

By M. W. VON BERNEWITZ

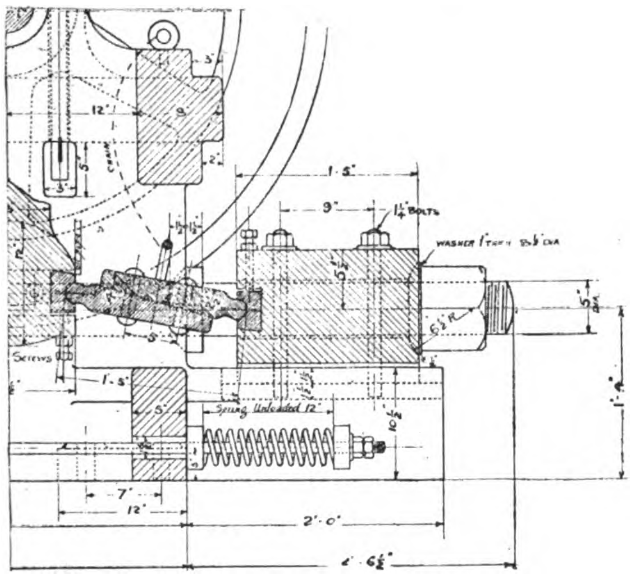
(October 12, 1912)

The relative merits of jaw and gyratory types of crushers have often been discussed in technical papers, so I shall only refer to the general practice of crushing ore, and the cost of this work at the large mines of Western Australia.

At one time it was customary to have crushers on top of the mill storage-bins; but now the practice is to have the crusher-house entirely separate from the mill. The broken ore is conveyed from the crusher to the mill-bins by an 18 or 20-in. belt-conveyor of either rubber or balata construction. Perched on the top of a mill, a crusher is a nuisance, as it shakes the whole building, repairs are difficult, and the distribution of ore is not easy. Crushers with the Bigelow patent toggle, in which 10 steel rivets, with a total shearing strain of 360 tons, are sheared off in case of an unbreakable substance getting between the jaws, are extensively used. This shearing is somewhat like the break-pin arrangement in the Gates breaker. Adjustment of the No. 7½ Gates, type K, is made at the top instead of at the step. The crushing head and spindle are suspended from this point, relieving the bottom of much weight. The spiders are set parallel to the flow of ore from the grizzlies, the ore thus fills the gyratory breaker all round the crushing head, tending to high capacity and efficiency. Motor-driven crushers are generally satisfactory, and circuit-breakers or fuses rarely blow out, although the feed is so irregular. The No. 5 and 7½ Gates take 30 and 65 hp., respectively, when full of ore; and the 30 by 12-in. jaw crushers take 30 hp. each.

Stage crushing is considered preferable at the large mines, including the Boulder, Horse-Shoe, Ivanhoe, and Perseverance. It is rather too much to expect any single breaker to crush 700 tons per day, considering the sizes in use and the number of hours at work. Some mines crush ore during the whole 24 hours; others dur-

ing two shifts; and others again during day shifts only. There was at one time an idea that motors would not do for driving crushers, as the sudden rush of ore into an empty machine caused a high flow of current which would open the circuit-breaker or other safeguard; but in practice this proves not to give trouble. The crusher at the Associated Northern, Ivanhoe, and Kalgurli are driven by steam engines. The former is driven from a counter-shaft, by belt from a clutch pulley on the main mill shafting. The Ivanhoe drives a countershaft by a small engine, the counter carrying rope pulley from



BIGELOW SHEARING TOGGLE

which three ropes drive a No. 7½ crusher, and two ropes each drive the two No. 5 Gates below—rather an unusual arrangement. The Associated, Boulder, Hainault, Horse-Shoe, Ivanhoe, Kalgurli, Lake View, and South Kalgurli crushers are fed directly from skips, the ore of course, first passing over grizzlies. The Associated Northern, one section of the Boulder plant, and Oroya Links crushers are fed from cars, while the Perseverance and one section of the Oroya Links are fed from a bin into which the skips are dumped.

I have seen the break-pin arrangement work properly on the No. 5 Gates, but the larger size is not provided with it. The Horse-Shoe had one crusher broken by an iron bar, while the Associated had two spiders broken, one hopelessly and the other cracked. A 2-in. band was shrunk around the cracked spider, while ½-in. boiler plate was studded on the arms, with good results. Recently a spalling hammer got jammed between the head and concaves, but the circuit-breaker, which is set at 135 amperes, flew out, and no damage was done. The table following gives details of the breakers at work at Kalgoorlie:

DETAILS OF CRUSHING PLANTS AT KALGOORLIE

Name.	Jaw.	Gyratory.	Type	Size	*Tons crushed per day.	How driven.	Cost per ton, cents.
Associated	..	1	Gates	K. 7½	360	80-hp. motor	10
Associated Northern	..	1	"	No. 5	70 to 120	engine	10
†Boulder	..	6	" Hadfields	No. 5 R. & S., 7½ and 5	600	motors	8
Hainault	3	..	Fraser & Chalmers	20 by 10 in. 16 by 9½ in.	230	"	10
Horse-Shoe	..	3	Gates	5 and 7½	800	"	..
Ivanhoe	..	3	"	7½ and 5	650	engine	6
Kalgurli	3	..	Fraser & Chalmers	30 by 12 in. 15 by 19 in.	360	"	8
Lake View	2	..	Bigelow	30 by 12 in.	600	motors	..
Perseverance	1	2	Hadfield & Gates	30 by 18 in. No. 3	700	"	11
Oroya Links	2	..	Bigelow	30 by 12 in.	360	"	10
South Kalgurli	1	..	"	30 by 12 in.	330	"	9

*Deduct 20% for fine.
†The Boulder also has 3 H-type Gates breakers for crushing ore to about 1-in, size for the Griffin mills.

In these costs the transport of broken ore to mill-bins is included; this item is about 4c. per ton.

It will be seen that there are 12 jaw and 16 gyratory crushers at work at the mines mentioned, and, so far as can be gathered, each type is giving satisfaction. When once supplied with new wearing parts, each machine works for a considerable time without any repairs. The life of jaws, mantles, and concaves varies considerably with the class of ore broken and the composition of the steel. Jaws last up to 180 days; while the No. 5 Gates liners will crush 100,000 tons. The No. 7½ Gates liners will run fully 270 days. Of the ore raised from the mines, about 20% is fine enough to pass the grizzlies. The hardest sulphide ore contains 65 to 75% silica. Manganese steel liners are quite satisfactory; but not cast iron, chilled for about ½-in. depth, as I saw one set replaced not long ago.

It is not necessary to fill in behind the jaws with metal, the bolts being sufficient to hold them in place; but of course a new mantle or set of concaves is run with zinc, as usual. Concaves are now supplied in two pieces, so that the bottom half, on which the bulk of the crushing falls, may be thrown out and another fitted in. In doing this, it is necessary to remove the top sections, to make the job easier. The wheel eccentric is generally changed at this time, although if poor metal is used for it, it will not last many weeks. One lasted 18 months in a No. 5 Gates and 9 months in a No. 7½. Plenty of good oil is essential for the gyratory, at the spider, where the spindle is suspended, as well as for the eccentric and bearings, and grease for the gear. The oil for the eccentric should be renewed every week, using some 5 gal. The bearings of the jaw crusher are lubricated with any good grease, squeezed in by the Stauffer system. The Perseverance casts the spiders and steel heads for the No. 3 Gates with good results.

Transport of or from crusher to mills is closely connected with crushing; in fact, costs generally embrace breakage and storage. At the Associated the crushed feeds on to the 18-in. inclined belt by a peculiarly-shaped chute, really a compound bend, whence it is elevated to a horizontal belt, which discharges the ore into either of two storage-bins holding 500 tons each. The Associated Northern distributing belt discharges into three separate bins. The crusher here stands on top of the mill-bins. The ore from the Edwards' crushing plant of the Boulder is taken by a 20-in. belt-conveyor to the mill, meeting the ore from the main shaft crusher, when part goes to the ball-mill bin and part to a trommel, the coarse from this being crushed in three H Gates crushers, set at about 1-in. opening; then all the fine ore from the trommel and the small breakers goes by belt to the Griffin mill-bin. The Horse-Shoe has two crushing stations, and from the storage-bins the ore is conveyed by rubber belts to the stamp-mill. The Ivanhoe conveyor is fed by Challenge feeders, and by an automatic tripper the ore is discharged into the stamp-mill bins. From the Kalgurli storage bin at the main shaft the ore is carried in ½-in. skips by a Bleichert tramway to the ball-mill bins. The Lake View broken ore is conveyed to the stamp-mill

by long inclined belts; that from the Star mine being transported in cars drawn by a small locomotive to the Lake View, where it is dumped down a chute, hoisted, and conveyed to the stamp-mill. The jaw crusher at the Perseverance is fed from a bin at the mouth of the main shaft, which is filled by the skips in the shaft and cars from No. 6 shaft, worked by a ropeway. The crusher discharges upon a belt which feeds two No. 3 Gates, the ore dropping in a 1000-ton capacity bin, from which a belt elevates it to the ball-mill bins. The Oroya Links ore comes from the Oroya mine by Bleichert aerial tram, and from the Eclipse mine in cars, holding three tons each, drawn by horses. All the broken ore is dumped into the stamp-mill bins by a tripper.

On the whole, the crushing plants on the 'Golden Mile' are worked cheaply, and give a minimum of trouble. Crushers get rough work to do, and the need constant attention, not necessarily repairs. The engineer's staff should examine them thoroughly the first thing every morning.

DRY v. WET CRUSHING AT KALGOORLIE

By M. W. VON BERNEWITZ

(March 15, 1913)

According to the last annual report of the Hainault Gold Mine, Ltd., owing to change in character of the ore being mined treatment by the stamp mill and cyanide plant is now unsatisfactory, and a dry-crushing plant may be necessary if the mine develops satisfactorially. The time seems opportune to review the treatment problem at Kalgoorlie. It is with some diffidence that I do this, athough after eight years' general observation one should be in a position to do so easily. There are so many points to study and compare that the discussion may easily be inconclusive. Having watched the treatment of the ores by the dry-crushing method most of this time, I may be accused of being biased in its favor; I have also studied the wet-crushing process, and the general remarks of those in charge have been forgotten.

In general, the ores are of a gray to greenish color, of a schistose structure, and they show an increase in hardness and silica content with depth. They contain pyrite and tellurides of gold and silver, but no minerals harmful to treatment. There is little free gold, it being mainly in the pyrite. The most important constituents have an average of SiO_2, 63%; Al_2O_3, 4%; $CaCO_3$, 12%; $MgCO_3$, 6%; FeO, %; and S, 3½ per cent.

When treatment of the Kalgoorlie sulphide ores was started, the ore was not understood, as nothing like it had been treated in Australia before. In 1899-1901 several large plants were erected, and, while results were low, yet they were the means of spurring metallurgists to further trials. Gradually the present satisfactory methods were evolved. In the dry process, the effect of roasting was not thoroughly understood, and the ore in the leaching vats

had to be cut out by picks and bars. Then the analyses revealed the changes in lime and magnesia during roasting. In the wet mills, losses through the gold in tellurides not being dissolved, and trouble through the pyrite not being properly saved, also upset results.

After the early failures in treatment, two schools of treatment grew up. The Associated Northern, Boulder, Kalgurli, Perseverance, and South Kalgurli now treat by dry crushing; the Hainault, Horse-Shoe, Ivanhoe, Lake View & Star, and Oroya Links by wet crushing. Machinery is as effective as can be, labor is well paid and intelligent, supplies of fuel and water are of the first order, economy is pushed into every department, and the plants are kept at work full time.

In the dry plants, if the roasting be good, there is little fear in subsequent operations, and there is only one product, and an easy one at that. It soon was found that tellurides were not soluble in ordinary cyanide solutions, but bromo-cyanide proved to be effective. It is used in wet mills where either concentrate, sand, and slime, or concentrate and all slime is produced. This necessitates three or at least two subsequent processes, and close supervision. Bromo is still used at the Horse-Shoe, Ivanhoe, and Lake View. The Oroya Links is not using it much now. Great care is necessary in its use, yet it has helped to materially increase the extraction in the wet mills.

The dry process is a trifle higher in cost than the wet process, but the residue from the former is lower than the latter. The comparison may be stated as follows: (1) dry process cost $2.50 plus the residue (average extraction 93% on $8.26 ore, equals 70c.) equals a total of $3.20 per ton; (2) wet process cost $2.24 plus residue (average extraction 88% on $6.84 ore, equals 69c.) equals a total of $3.20 per ton. These figures are the best obtainable, as all cost systems are not similar, and extractions are not always published. It is admitted generally that the dry work is superior to the wet. During the month of May last, the tonnage treated dry divided into the total yield gave the above average figure. In the same month, wet tonnage and output gave the average quoted.

On the one hand there is a residue from the dry plants which will not pay for re-treatment; but from the wet mills a residue of $1 or over can be re-treated by vacuum-filtration at a cost of 50c. per ton; and if the loss in treatment be 20c., the profit would be 30c., which is worth saving. A few of the dumps contain more than $1 per ton, but I speak of results of present practice. The Oroya Links published figures giving the value of its residue lately as 42c. Against the first cost plus residue of $3.20 in the wet mills, must be placed cost plus residue plus re-treatment in the wet process, or $3.20 plus 50c. equals $3.70 per ton. This shows an advantage of 50c. in favor of the dry process.

Millmen maintain that a deal of gold is lost in dry crushing and roasting. From ball-mills and dust-houses about 0.5% of the ore may be lost, and in roasting in a modern furnace not more

than the same amount. Dust losses are difficult and costly to determine, but estimating all dust to average in value with the original ore, one per cent of 2400 tons dry treatment daily at $8.26 ore equals 8c. per ton lost. Experiments have proved that there is no actual volatilization of the gold in roasting telluride ores. The average loss in weight in crushing and roasting ore at Kalgoorlie is 10%; made up of dust, 1%; moistrue, 1%; sulphur, 2.5%; and CO_2 from carbonates, 5.5%. In roasting concentrate, the loss is only about 0.05%, and as this product averages some $50 per ton, the loss on 7000 tons treated monthly would be 3c., or 0.25c. per ton on 2730 tons treated wet daily.

The only amalgam in a dry plant is in the pans, while in the other process amalgam is collected from boxes, plates, pans, in battery, and the concentrate plant. There is accordingly more supervision necessary in a wet mill, and risk of stealing is greater. The average loss of mercury is low.

Concentration has been carried to a fine point at Kalgoorlie, but none the less very fine mineral and telluride floats and mixes with the pulp going to leaching vats and slime plants. This fact is, to my mind, one of the reasons for difficulties in a wet plant on such ores, in spite of the use of bromo-cyanide.

Classifying in a dry plant consists only in separating the pulp for the pans, and any coarse discharge from the latter is returned for further grinding. In some stamp-mills, classification for tables, re-classification for tube-mills, and further classification for more tables, is necessary, and the system rapidly becomes complicated. It necessitates extensive systems of classifiers, launders, and return pumps.

The average cost of ball-milling is 50c., against say 42c. per ton for stamping. I think the former plant easier to look after than the latter. Both produce products that are equally well adapted to further treatment.

This discussion would not be complete without mention of the general efficiency attained in roasting ore or concentrate. For capacity, fuel consumption, power, labor, and results, furnace work at Kalgoorlie cannot be beaten. As examples of roasting ore, the plants at the Associated, Perseverance, and Kalgurli may be mentioned, and of roasting concentrate at the Horse-Shoe.

One of the troubles at Kalgoorlie is the presence of graphite, which is found in the Associated, Associated Northern, Boulder, and Oroya mines. It does not seem to change during roasting, and causes a premature precipitation of the gold from cyanide solutions. Fortunately, the mineral does not come to the mills very often; but as to this effect, there is no doubt, and it is a nuisance hard to deal with. The Boulder is troubled with it more now than any of the other mines.

Gradually the filter-press is being discarded in favor of vacuum-filters. In the past an immense sum has been spent on press plants, about 100 presses being erected, 80 of which are treating 100,000 tons monthly. The balance are out of commission, but

the benefit derived from this machine at Kalgoorlie has been admittedly large. With a press washing can be carried to a degree that cannot be beaten, but the labor cost is high, and I expect that it will have to give way. The Associated Northern, Boulder, and Oroya Links filter their current mill slime by vacuum systems of their own, while others are talking about introducing the system. The lowest cost of treatment by filter-press is 24c. per ton, while 8c. should cover vacuum-filtration. Such a saving is important with ore yielding as low as $5.50 per ton.

The cheap price made for water, 36c. per 1000 gal., for sluicing away all mill residue, has simplified this work and done away with the further building of dusty dumps. Sluicing, on the whole, is cheaper than making dumps, and Kalgoorlie has much to be thankful for in connection with the great water scheme.

On the cost of erecting the two types of plants discussed, there is not much data on account of constant additions, improvements, and alterations. The Associated Northern plant, with capacity of 120 tons of sulphide ore daily, cost originally $180,000. For the same tonnage, a battery of 25 stamps would cost $125,000.

The margin between the dry and wet processes was large enough at one time, although extractions vary from 94 to 86%, respectively, and left no doubt as to which should be chosen. Now the stamp-mill work is improving slowly but surely, and the difference is small. For my part, I would build a dry-crushing and roasting plant to treat a similar ore.

HEAVY STAMPS

(*The Mining Magazine,* June, 1910)

E. J. Way, consulting-engineer to the New Kleinfortein, in the May *Bulletin* of the Institution of Mining and Metallurgy contributes a criticism of W. A. Caldecott's paper on the development of heavy stamps. He shows that Mr. Caldecott does not make out a case for the greater crushing efficiency of the heavy stamp, but only for a greater capacity. Indeed, seeing that in Mr. Caldecott's experiments the heavy stamp used more water per ton of ore and the proportion of +60 was increased, there was really a decrease of efficiency. Mr. Way also combats Mr. Caldecott's four claims for the heavy stamps relating to the general decrease of initial and current costs, and points out that the first cost of foundations and constructional work is far greater with heavy stamps than with light stamps, and that the increased size of the ore-bins for heavy stamps compensates, as regards space occupied, for the smaller area covered by the stamps. In calculating first cost allowance must also be made for the additional plant required in the tube-mill circuit and for the buildings in which it is housed. As regards the cost of running, Mr. Way considers that the expense for wear and tear will adversely affect the bill in the case of heavy stamps, while the increased cost of supervision in con-

nection with the tube-mills will more than counteract the lower cost of labor per ton in connection with the heavy stamps.

Mr. Way points out that the results obtained by the Mines Trials' Committee elucidate the question of the efficiency of the heavy stamp. This committee found that the maximum mechanical efficiency when using heavy stamps is only obtained by crushing through correspondingly coarse screens; in other words, for every mesh there is a corresponding stamp-weight which would secure the highest efficiency. From this point of view the general aspect of the question changes. The reason for the two-stage crushing is really that with large and heavy stamps efficiency can only be obtained by the adoption of unusually coarse crushing. Mr. Caldecott stated that "the heavier the stamp the coarser the preliminary breaking admissable and *vice versa.*" Mr. Way, expanding the last two words thus: "the lighter the stamp the finer the preliminary breaking admissable," considers this proposition the basis for future guidance and urges that the function of a heavy stamp should be to prepare the ore for reduction and amalgamation in light stamps. This plan would be in consonance with the results obtained by the Mines Trials Committee. Under these circumstances the economic limit of weight of the heavy stamp, leaving mechanical difficulties out of consideration, would be reached only when they encroach on the province of the jaw-breaker or gyratory crusher. Even with the coarsest crushing by heavy stamps there is a substantial proportion of minus 90 formed; and the lighter stamps following the heavier would be of smaller total capacity seeing that some of the ore, being already fine enough, would be eliminated.

Mr. Way is not convinced with regard to the claim that the recent fall in working costs on the Rand is due to the adoption of the tube-mill. His experience goes to show that the lowering of the cost of treatment is due to the increase in the scale of operations and the cheapening of labor and supplies. At the New Kleinfontein, Mr. Way has never adopted tube-mills, and yet his reduction expenses are lower than most of the double-stage plants. The difficulty of apportioning the cost, owing to the variety of methods of accounting renders it impossible to give an accurate comparison. Nevertheless the table of costs at 30 mills on the Rand presented by Mr. Way is sufficient to show that he has abundant arguments on his side.

CRUSHING BY STAGES

(July 16, 1910)

The Editor:

Sir—The suggestion made editorially in the current issue of *The Mining Magazine,* based on the views held by Edward J. Way, that a combination of heavy and light stamps should prove to be advisable, seems to me a valuable one, and it would not

surprise me to see, before long, not only a combination of two, but possibly also of three, and, in exceptional cases, more sizes in operation, to be followed by tube-mill grinding. The modern practice of comminution has been a source of astonishment to me for many years, believing, as he does, that the weight of the tool should be proportioned to the size of the material. To grind rock down to 40-mesh with a 100-lb. stamp, or a 1000-lb. muller, seems as unscientific as it would be to cut a watch wheel in a 48-in. lathe. There is no getting around the fact that for correct grinding it is necessary to select the weight of the hammer, or rather its energy, so that it is just sufficient to crush the particles to be ground, and, as a corrolary, that all particles smaller than those suitable for each machine be removed before admission to it. This principle is interestingly illustrated in the construction of the Hardinge conical tube-mill (to use this, apparently unavoidable, misnomer), the ingenuity and efficacy of which the mining fraternity is so slow in recognizing. To what extent these principles can be carried economically into execution can only be determined in each case individually, but it is safe to say that they are capable of application to advantage in almost every existing plant, and the nearer approach to perfection in design and workmanship of machinery, and the greater in experience gained in connecting and operating the independent units, the more of the latter will it be profitable to use, thereby coming closer to the theoretical efficiency. One of the reasons why the stamp-mill, in spite of its faults, has so long maintained its supremacy in competition with other milling devices, is because of its superiority in discharging the ground material. If, then, stamps were chosen so that their weight did not exceed that required to accomplish the breaking of the ore, they would make as ecomonical a crushing machine as can be expected for the smallest sizes, where stamps would become impracticable, and tube-mills, preferably of the conical shape, would take their place.

Fundamentally the stamp which automatically stores a certain energy and re-delivers the same at each blow, regardless of the power required, may not be an ideal machine, but its simplicity of construction and operation will probably assure it the foremost place among medium-sized crushing devices, provided it is adapted to the work to be performed on the lines that have been suggested.

F. CREMER.

Los Angeles, July 6.

(July 30, 1910)

The Editor:

Sir—Most millmen have practiced crushing by stages, but not in stamp-mills alone. Every machine has its sphere of usefulness, or its limitations. The rock-breaker of different sizes, for breaking ore between certain sizes, depending upon the physical character of the ore; the stamp-mill for other sizes and the pulping machines, such as Chilean or Huntington and tube-mills for still other sizes.

Stamps of weights between 850 and 2000 lbs. approach at both ends the limit where the rock-breaker or the fine-grinder is more efficient. The 2000-lb. stamp approaches the rock-breaker, while the 850-lb. stamp comes near the sliming machine. While the complication of machinery would no doubt be profitable in very large mills, for the average small mill it would be out of the question. That portion of the mill where the most expensive grinding takes place (the fine grinding plant) is where this principle might be better applied.

F. Cremer says the mining fraternity is slow in recognizing the efficiency of the principle as illustrated in the conical tube-mill. Maybe the mining fraternity has found the weak points of the conical tube-mill and prefers two cylindrical tubes, sizing between, or crushing in stages instead of in one machine. Mr. Cremer will do a favor if he will point out in what particular the conical tube-mill is more efficient than cylindrical tubes. My idea is that the conical tube has its limitations, just as the stamp-mill has, and for fine grinding, from 100 to 200-mesh, the cylindrical machines properly proportioned are superior.

If in a 5-stamp battery each of the stamps were of different weight and ran at different height of drop and different speed, we would have the same principle as in the conical tube-mill, less the sizing action. The heavy stamps would do most of the work, as is the case in the larger diameter of the conical tube. The question is, to what extent does this sizing action take place. We know that on dry ore with no discharge of ore from the mill, the sizing action of pebbles and ore particles is pronounced, but how far does this take place in wet crushing, where a continuous stream is entering and leaving the mill? The pebbles size, of course, but does the ore size, or do the small pebbles have to work on large particles of ore? If so, the principal that Mr. Cremer favors is not found in the conical tube. My experience is that this sizing action on the ore is very imperfect and consequently the small pebble, with its comparatively small amplitude of fall, is in the position of an 850-lb. stamp, taking the size of ore that had better go to the rock-breaker.

ALGERNON DEL MAR.

Fort Bidwell, California, July 20.

(August 20, 1910)

The Editor:

Sir—It is with some reluctance that I enter into a controversy with so experienced a writer as Algernon Del Mar, especially on a question that cannot be anything but a hypothetical one at best. The so-called 'efficiency of grinding machines, as of so many other industrial contrivances, is very uncertain, dependent as it it on many elusive factors that nobody can control. If this be true in regard to so widely divergent types as jaw-breakers and gyrating crushers, stamp-mills, and Chilean mills, and of the large number of varieties within the latter class, where the preference for a cer-

tain machine develops simply into a question of local conditions, and often personal inclinations and prejudices, how much more difficult, if not impossible, must it be to decide as to the superiority between so closely allied methods as the ones under consideration, namely, grinding in two cylindrical tube-mills, with an intermediate sizing of the product as preferred by Mr. Del Mar, as against grinding in a conical mill alone. The latter, in my estimation, has not been given the proper opportunity to prove its merits (or weaknesses), and Mr. Del Mar himself takes his stand against it on entirely hypothetical grounds. At least he produces no concrete evidence for his preference, and I doubt whether the results upon which he bases his opinion have been recorded with that degree of accuracy necessary to preclude any possible error of judgment.

To do so would be impossible, except with the most scrupulous attention to details, as was demonstrated by the recent drilling contest on the Rand, where the most elaborate precautions had been taken to prevent any bias and the results were recorded with the utmost care. Nevertheless, the decision, in favor of a certain drill, was so unsatisfactory as to call for a second competition, and with so disappointing a result that even the S. A. Chamber of Mines, under whose auspices, together with the Transvaal Government, the trials were held, considers them of little importance, while the Johannesburg correspondent of *The Mining Magazine* pronounces them void of all finality and capable of yielding entirely different results if the contest were to be conducted again under the identical conditions. If this is the result of one of the most scientifically organized tests of modern times—as far as efficiency of mining machinery under actual working conditions is concerned—and if this agrees with the experience we gain every day when we attempt to condemn a machine or method on the strength of an apparent failure under certain conditions, I wonder at the courage of Mr. Del Mar in assuming the rôle of supreme judge when he declares that "the fraternity may have found the weak points of the conical mill and prefers two cylindrical tubes, sizing between, or crushing in two stages instead of in one machine."

That I perfectly agree with him in regard to the value of crushing in stages is made clear in my first remarks (the ones that gave so much offense), and for that reason I consider the principle of the conical mill correct, inasmuch as it is certainly better to perform the work in one machine, when possible, instead of in three. If in actual operation the results should not correspond to the principle involved, after exhausting all possible means to reconcile both, then the machine will be abondoned altogether, or relegated to its proper sphere. That this stage has not been reached yet, I am free to say, and all the more so as there are authorities who favor the conical mill and large enterprises that are perfectly satisfied with its operation. Personally, I have no evidence to offer, either way; and if I had, I would not consider it conclusive. The fact is that I have never given this matter much attention and was therefore

surprised to see Mr. Del Mar make my inconsequential remarks the subject of attack. I still maintain that the principle upon which the conical mill is built has not been sufficiently recognized by the mining fraternity, as is shown by the small number of installations that have been made. Mr. Del Mar tries to prove the inability of the conical mill to accomplish the desired result by doubting that much selective action takes place on the part of the pebbles, and claims that he has learned this from his experience. It would be interesting to know wherein this experience consists, because to me it would seem more plausible to expect considerable selective action owing to the spherical form of the pebbles, which should give the fine particles in the pulp an opportunity to escape the impact between the fewer large pebbles near the inlet (but catching the larger particles) and submitting them to the crushing action of the more closely packed small pebbles near the discharge end. Mr. Del Mar himself admits a 'pronounced' sizing action in the case of dry ore. If so, what right have we to assume a *priori* that all, or nearly all, such action ceases once water is introduced? That the greater fluidity of the water must have some effect will probably be admitted, but in what direction, and to what extent, remains entirely a matter of speculation.

Mr. Del Mar's comparison of the conical-mill principle with that in a battery of stamps of different weights appears to me quite irrelevant. An analogy might be claimed if the ore entered the battery at the end of the heaviest stamp and left at the opposite end, but even then the points of similarity are so few, and those of difference so many, that their comparison seems hazardous. From Mr. Del Mar's uncompromising attitude toward the conical mill one would feel inclined to suspect that his trial with the same must have been disastrous (possibly due to the mill—possibly not). Let it be hoped that such is not the case, and that his rush to the defence of the 'fraternity,' as well as his somewhat peremptory challenge to me, are not the reaction of a pardonable state of mind under such conditions, but his unmixed desire to dispel, in an entirely disinterested manner, any favorable impression in behalf of the conical mill my remarks may have created, because contrary to the truth, as he sees it.

Before closing I should like to state that the omission of the word 'except' somewhat obscured the meaning in my last communication. The last sentence of the last paragraph but one should read as follows:

"If we, then, choose our stamps so that their weight does not exceed that required to accomplish the breaking of the ore, we should have as economical a crushing machine as we can expect, except for the smallest sizes, where stamps would become impracticable, and tube-mills, preferably the conical variety, would take their place."

FELIX CREMER.

Needles, California, August 7.

The Editor: (August 20, 1910)

Sir—There are one or two items that might be added to the discussion on stage-crushing, which has recently appeared in the columns of the *Mining and Scientific Press.* Without taking exception to anything that has been said previously we may safely say that the question is a matter of cost and that efficiency is only of value to the extent that it affects cost. The plant that everyone is striving for is the one in which the treatment cost (including amortization charges) plus value in the tailing shall be the minimum. Bearing this point in mind, the size of the mill and the amount of ore on hand will usually have a greater influence on the selection of equipment than the differences in efficiency that stage-crushing may produce. The largest plants, which must in any case have a large installation of machinery, can look more closely at the advantages of stage-crushing than others. Again, the mechanical problem of transferring the pulp through a long series of machines is a serious one. Either several feet of that never too plentiful mill height must be sacrificed or some sort of elevator must be put in. In the case of stamps, a departure from the usual arrangement, in a single row would be objectionable. An example of how stage-crushing may be carried too far is furnished by a mill equipped with a large crusher discharging to a trommel whose oversize goes to a short head crusher. No grizzly was used ahead of the first crusher. Had a grizzly been used and the first crusher set up one inch, an elevator, trommel, and crusher could have been eliminated and the size of the building materially reduced. The capacity would still have been ample so that one shift could supply the mill for a day. It is hard to see how stage-crushing pays in such a case.

The 'removal of the fines as soon as made,' so frequently urged upon metallurgists, is no child's play, and although theoretically correct, it must not be done until it can be done at a profit. It would seem that each case is a separate problem whose answer reads in dollars rather than in percentages.

L. B. EAMES.

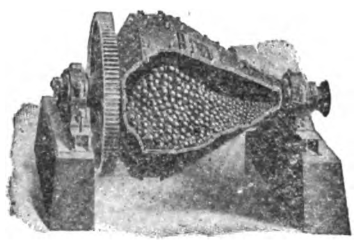
Pueblo, Colorado, August 9.

The Editor: (October 8, 1910)

Sir—As the indirect cause of a most animated discussion as to the real and supposed merits of the range and efficiency, or if you will, the inefficiency or inability of the conical pebble-mill to fill all gaps and delinquencies of all other fine-grinding machines to which the metallurgist falls heir, I have read with a parent's interest the discussion that has been going on for some weeks in your issues, between Felix Cremer and Algernon Del Mar. I would hardly be human did I not feel kindly toward Mr. Cremer in his advocacy of my 'baby.' He frankly states that the child looks healthy, pats him on the back and encourages him to win a place among the grown-ups, while Mr. Del Mar takes the position that even though his body is formed on correct lines, he does not be-

lieve that he will be able to hold his own. Then comes Stuart Todd into the discussion, in which he refers to a Hardinge conical mill as a single-coned mill. Every conical mill, so far built by my company, was and is double coned. Some time since we felt highly flattered, though we must confess a little annoyed, to find that our child was attracting the attention of the kidnaper. We now allow him to run the streets alone, for he has gained so many powerful friends that we have no longer fears for his safety.

Mr. Del Mar is right when he says that he believes the conical mill has its limitations. In this 'it is almost human,' at the same time we believe he is wrong in advocating two tube-mills and intermediate separation, as preferable to such 'stage reduction' as is obtained in the conical mill, for the reason that he uses the same mediums for further finer division of particles which have already divided them in the first mill. The principle of ecomony involved in all reduction is to successively reduce the energy as the particle is divided, for the energy necessary to divide a mass of matter is directly proportional to the particles of the mass, hence each



HARDINGE CONICAL MILL

division will require the application of less force. Again we will agree with Mr. Del Mar that stamps of 2000 lb. are encroaching upon the work of the rock-crusher, and one of 850 lb., that of the sliming machine. The 'stamps' we would advocate, based upon our experience during the past few years, should begin with a ball of 12 to 15 lb. weight acting directly upon the rock-crusher product and pass the resulting mass to gradually reduced weights down to one pound, or less. By figures, it will be ascertained that this one-pound ball has more effective action upon a 12-mesh particle than the 2000-lb. stamp upon a 2-in. cube of quartz; the latter system has a weight relation of about 2500 to 1, while the former is more than 20,000 to 1. It is the step reduction, such as this, that underlies the principles involved in the conical mill.

The real efficiency of any step reduction will be in direct proportion to the hard crushing surfaces exposed. Crushing with stamps involves the principle which confronted every metallurgist in his first ore reduction effort on the bucking-board. He will remember how he constantly cleaned the board, sieved and returned the coarse material for further reduction—not that he was seeking out any great principle, but simply knew he was expending unnecessary energy for which he was paying. After a few vigorous

rubs, and a few drops of 'elbow grease'—from his brow—he sought the course of least resistance by cleaning his bucking-board and presenting a clean, hard surface for final reduction. Again we must agree with Mr. Del Mar, in theory, but in practice it would work about as follows: The stamp exposes crushing surfaces of shoe and die of about 60 sq. in. in each, a total of say 120 sq. in., or in a stamp of 2000 lb. about 16 lb. of weight for each square inch, this stamp with a 6-in. clear drop above a 2-in. cube of quartz, will (provided the die is clear of other material), easily reduce this size of cube to a fractured mass, the average of which will be finer than particles or cubes of less than ¼ in.; a reduction of 512 to 1. Reasoning upon this line, and as stated above, crushing resistance being proportionate to the particles of the mass, the next stamp in the series would be approximately 1/500 of the original, or a 4-lb. stamp. This we propose to furnish in a 3-in. ball with a crushing surface of 27 square inches, or 7 sq. in. of crushing surface for each pound of weight, or relatively 13,000 times that of the stamp, to say nothing of the proportionately increased amplitude of drop, which in the case of the '4-lb. stamp' again multiplies its efficiency many times. This relation of stamps to the multiplicity of balls, the decrease in the weight of the stamps to the decreasing weight of balls, and height of drop of stamp, offset by amplitude of fall or roll of balls, is illustrated in the following cut of a conical mill and a battery of stamps of different weights.

Even stamp advocates it would seem, must admit that ideal step-reduction would be successively reduced weight of stamps to crush material proportionately reduced. Even the smallest stamp illustrated, if it weighed but 100 lb., a ratio of 1:10, with the 1000-lb. stamp, would have an increased relation to the size of material to be reduced in its succession, as the fourth power of 500, or millions of times greater than the original 2000-lb. stamp to its 2-in. cube. Thus, it will be seen that even the practice obtained in the conical mill does not come anywhere near its theoretical efficiency owing to the interference of the already finely comminuted particles. At the same time it is a long step in the direction of automatic adjustment of power to work performed. We are of the opinion that this matter of exposure of clean, hard, crushing surface, deserves careful consideration by the mill-man, especially as metallurgy through the production and utilization of low-grade ores, is assuming a position of industrial manufacturing importance, in contradistinction to the uncertain occurrence of high-grade ores with which mining has generally been associated. We are acquainted with an instance where an additional saving of $^1/_{10}$ of 1% of copper content per ton is the basis for reconstruction of a plant involving the expenditure of a very large sum of money. This saving of an additional $^1/_{10}$ of 1% seems small, yet in the case under consideration means a net gain of over $500,000 per annum at a lesser net cost than that at present.

H. W. HARDINGE.

New York, September 30.

(November 5, 1910)

The Editor:

Sir—In your issue of July 30 I took exception to Felix Cremer's remarks in regard to the mining fraternity not recognizing the efficiency of the principle of grinding as illustrated in the conical tube-mill. I made some criticism of the principle involved mainly for the purpose of getting accurate data from metallurgists who have the two types of machines, cylindrical and conical, working side by side. The results on which I based my opinion were, as Mr. Cremer suggested, not recorded with sufficient accurary to preclude errors of judgment and as the two types of machines were not used, no comparison was available. Mr. Cremer has offered no figures nor any mathematical deductions to prove his point.

Then comes Stuart Todd, who supplies concrete figures of an actual comparison of two mills working side by side on the same ore. This is the kind of information that is wanted. The comparative efficiency of the two mills is herein worked out. H. W. Hardinge, the professed parent of his 'baby,' as he calls it, enters into the discussion, October 8, but offers no figures of actual work done. I am surprised at some of the mathematics he has offered. I was unaware that all of the surface of the balls introduced into a tube-mill was effective crushing surface, for I had labored under the delusion that the balls only touched at points, and not over their entire superficial area, which renders Mr. Hardinge's mill '13,000 times' more effective than stamps of a weight equivalent to the weight of balls in his conical mill. Doubtless Mr. Hardinge has authenticated figures which represent the actual (not theoretical) performance of his mills, and if these were published much needed light would be thrown on the subject. I notice Mr. Hardinge takes exception to Mr. Todd calling his mill a single-cone mill. This is unfortunate, for if the principle involved is correct there should be a central feed and double discharge, as Mr. Abbé has developed his double conical mill. Why? Because if this sizing action is pronounced the tendency must be to keep the fine material at the feed end and to push the coarser material in the opposite direction—forcing this coarse material to that portion of the cone where the crushing efficiency is at its lowest. In other words, the heavy stamp has missed the coarse ore and the light stamp must attempt to do this work at a disadvantage. (Note Mr. Hardinge's comparative illustration of stamps in the tube-mill.) This point is proved by Mr. Todd's figures, 15.1% + 20-mesh, the feed, and 3.5% + 20-mesh, the discharge, showing that 3.5% of the ore passed entirely through the mill without any reduction in size.

I do not agree with Mr. Hardinge that I take the position that the body of his mill "is formed on correct lines." I believe the nearer it approaches the cylindrical the more effective it would be, but I did suggest in *The Mining World* that provided the sizing action was pronounced, as claimed, that an ellipsoidal-shaped mill would be more effective than the conical because it gave more

'large diameter.' A mill, to be effective, must revolve at a speed proportionate to its diameter, or, as the formula shows, inversely as the square of the diameter, the variation in speed due to load of pebbles, physical quality of ore, etc., being secondary. A conical tube cannot conform to this condition, because all diameters make a revolution in the same time, but not at the same peripheral speed.

I have recommended two mills or more, and intermediate sizing not, as Mr. Hardinge states, using the same media, but mills of different diameters, different size pebbles, and different speeds. This is a different proposition. To analyze Mr. Todd's figures, it may be supposed that the mean grade of ore remaining on a 20-mesh screen to be that corresponding to the average of the sizes between 10 and 20 mesh, that on a 40-mesh to be an average between 20 and 40, and so on, and that going through 120-mesh as the mean between 120 and 200. The figures will be as fair for one machine as for another. The sizes of the apertures in the different mesh screens may be taken as follows, calculated as in Richards' 'Ore Dressing.'

+ 10 mesh	0.0205 inch
+ 20 "	0.027 "
+ 40 "	0.0115 "
+ 60 "	0.0072 "
+ 80 "	0.0057 "
+ 100 "	0.005 "
+ 120 "	0.004 "
+ 150 "	0.0036 "
+ 200 "	0.003 "

"The problem becomes the mathematical one of determining the relative surface of the grains of pulp before and after passing through the tube-mill, assuming similar shapes and equal weights of all the particles included in each successive grade. Since for any definite weight of ore the total surface area varies inversely as the diameter, then the relative surface exposed by each grade may be determined by dividing the weight of that grade by the average diameter of the particles in the grade."

COMPARATIVE EFFICIENCY TABLE

Cylindrical Tube

		Feed.		Issue.	
Mesh.	Mean grade.	%	Relative surface.	%	Relative surface.
+ 20	0.023	3.1	1.34	....	....
+ 40	0.019	15.5	8.15	3.2	1.63
+ 60	0.009	15.6	17.33	4.5	5.00
+ 80	0.006	14.6	24.33	10.5	17.50
+ 100	0.005	9.6	19.20	15.8	31.60
+ 120	0.0045	2.4	5.33	2.4	5.33
— 120	0.0035	39.2	112.00	63.6	181.71
			187.68		242.77
					187.68
					55.09

Conical Tube.

Mesh.	Mean grade.	Feed. %	Feed. Relative surface.	Issue. %	Issue. Relative surface.
+ 20	.... 0.023	15.1	6.56	3.5	1.51
+ 40	.... 0.019	55.6	18.78	21.2	11.15
+ 60	.... 0.009	10.4	11.55	14.1	15.66
+ 80	.... 0.006	11.1	18.50	17.9	29.83
+ 100	.... 0.005	6.4	12.80	11.2	22.40
+ 120	.... 0.0045	0.7	1.55	1.6	3.55
— 120	.... 0.0035	20.7	59.14	30.0	87.14
			128.83		171.24
					128.83
					42.41

I have divided the product by 100 to bring the calculations to unity. The relative mechanical efficiency is obtained by dividing the work done by the amount of power used. The cylindrical tube has reduced the surface 55%; the conical tube 42.4%. If the power and tonnage were the same the relative work done would be as 55 to 42, but while the tonnage is the same the power required is different.

$$\text{Work done, conical, } \frac{55 \times 75}{30 \text{ hp.}} = 127.2$$

$$\text{Work done, cylindrical, } \frac{42.4 \times 75}{25 \text{ hp.}} = 137.5$$

This shows the relative efficiency of the cylindrical and conical mills as calculated from Mr. Todd's figures of actual work accomplished, to be as 137.5 to 127.2.

ALGERNON DEL MAR.

Fort Bidwell, California, October 15.

STAMP-DUTY ON THE RAND

(Editorial, October 22, 1910)

For years past the stamp-mills of the Rand have led the world in output per stamp. Originally crushing from five to six tons per stamp each twenty-four hours, coarser screens and heavier stamps were introduced to increase the stamp-duty until it reached nine or more tons per stamp daily. Recently, at the East Rand Proprietary, experiments have been made of a radical nature, which may greatly change metallurgical methods, by vastly increasing the tonnage crushed per stamp. These experiments have been carried on at the Cason and Angelo mills, and it is stated present indications are that the stamp-duty will be increased to twenty-five tons daily per stamp. The number of tube-mills is to be increased to one for each ten stamps. These mills have 220 stamps each, and will therefore require, when fully equipped, forty-four tube-mills. It is contended that by this method of operating—crushing with

stamps through three-eighths inch mesh, and sending this product to tube-mills, at the rate of twenty-five tons daily per stamp—the expense of power will be reduced fully fifty per cent. At this rate each of these mills will have a crushing capacity of 165,000 tons monthly, or a total for both of 330,000 tons per month. This, however, is considered as beyond the present capacity of the mines, which, by the way, comprise a very large area, but it is proposed to install a sufficient number of tube-mills to bring the output up to 200,000 tons monthly, which would be equivalent to over 6600 tons every twenty-four hours, and would be the largest tonnage output of any gold mine in the world, exceeding the tremendous output of the Homestake of South Dakota by 2600 tons daily.

An interesting problem is presented in contemplation of the ambitious plans of this Transvaal mine management, and that is the ability of the mines to supply this large tonnage continuously for a considerable period. At the beginning of 1910 the East Rand Proprietary Company owned 4312 claims, and early in the year opened negotiations with the Transvaal Government for the right to mine under 169 additional claims, consisting of township, water-right, and other lands. At the beginning of the year ore reserves were estimated at over 11,000,000 tons, having an average value of $5.75 per ton. The width of reef is variable, but as nearly as can be determined from consulting the latest authority on mining in the Transvaal, Mines of Africa, the width is from 24.4 inches to 29.2 inches, on which the total development exceeds 84 miles. To December 31, 1909, the total tonnage extracted had exceeded 7,928,000 tons. In any event, notwithstanding the great claim area, extensive development and vast ore reserves available, the constant production of 6600 tons of ore daily, as contemplated, is a proposition which will require all the skill, energy, and good fortune which may fall to their lot, but the Rand is a country of large possibilities.

PROGRESSIVE MILL PRACTICE

(Editorial, January 21, 1911)

California is often referred to as the cradle of gold mining, and such, indeed, it really is, but it is a matter of general surprise that there is a tendency in some directions, in California, to remain in the cradle. Particularly is this noticeable in the treatment of gold ores of the State. Other gold-mining regions throughout the world have in their early history, almost without exception, adopted California mill practices, but in many cases the newer countries have promptly drifted away from the 'time-honored customs' of California's millmen and evolved newer and better methods through experimentation, or have applied those already tried out in other new regions. In no district is this departure from traditional methods more radical than on the Rand. The ores of the Rand are relatively simple—a quartzose gangue with metallic gold and auriferous pyrite. The ore yields its gold readily to amalgamation

and cyanidation. As concentration is not considered necessary on the Rand, the engineers, to reduce cost, years ago began to seek for methods of milling which would permit an increase of stamp duty. Year after year saw the stamp-duty increased by various ingenious means, until now, by coarse crushing and re-grinding, the capacity per stamp, as at the East Rand mills, for instance, has reached 20 tons daily, and even a higher rate is anticipated, as compared with that of 4 to 6 tons per stamp in the average California mill. That the gold ores of California are mostly readily amenable to the simple methods of recovery practiced on the Rand is well known, and why California millmen do not make an effort to increase mill capacity along lines similar to those that have been successfully evolved on the Rand, is one of the things not easy to understand. An increase of stamp-duty without any material increase in the cost of power will certainly result in a decrease of milling cost per ton. At many California mines, by adopting the re-grinding methods of the Rand, employing Chilean or tube-mills after coarse-crushing in the stamp battery, the capacity could be raised from 4 or 5 tons per stamp to 10, and possibly to 15 tons, per stamp daily, while making as large a saving as by present methods. It seems well worth the effort, at any rate, and if the attempt be successful it would permit a stated capacity at a much lower capital expense for equipment.

(February 25, 1911)

The Editor:

Sir—In your issue of January 21, under the heading of 'Progressive Mill Practice,' you suggest that California millmen be more progressive and alert to the possibilities developed by others working along the same lines, referring more particularly to the practice in South Africa of equipping the stamp-mill mortars with coarse screens and thereby increasing the capacity of the plant to about double, without a proportionate increase of power consumption.

If you will refer to the article by myself in the *Mining and Scientific Press* of April 30, 1910, you will see that at least one California millman is not so slow in the recognition of a good thing, and was more of an originator than imitator of the practice which is being so almost universally adopted in South Africa.

"At Tuolumne Mr. Elmer is crushing through a four mesh battery screen. A short copper plate is interposed between the screen and the pebble-mill and the total product, amounting to upward of about 100 tons in 24 hours from 10 stamps, is then passed directly to an 8-ft. Hardinge mill into which is fed at regular intervals of half an hour, a spoonful of quicksilver; the regulation wooden mustard spoon being used. From the pebble-mill the pulp is run through a box distributer over a second set of amalgamating plates, where additional free gold or amalgam produced within the mill is caught." (A plan of the flow-sheet then follows.)

This innovation by Mr. Elmer was based upon the use of coarse screens on the stamp batteries of the Creston-Colorado mine at Minas Prietas, Mexico. Coarse screens were adopted there, to my certain knowledge, as far back as 1897, and have been continuously used at that mine, with the exception, however, that they are now using screens of practically ¾-in. mesh, the discharge going to re-grinding machines.

Though admitting that the California millman is practically wedded to his stamp, at the same time full credit must be given him for setting a style when he furnished his lady with a new dress which does not 'hobble' her activities. The introduction of coarse crushing by stamps was the cause of my being such a staunch advocate of the oft-repeated statement that the stamp should not be considered as a fine grinding machine, for as such it is wasteful of energy.

Our friends in South Africa deserve all the credit they are receiving for what to them is original work, but at the same time we should give credit to any young worker who makes an advance in methods, such as Mr. Elmer did, in a field of conservative stamp-mill practice like those found in many of our Western mining camps. For myself, I believe that the stamp has seen its best life; I realized this in 1897. Then why does it exist? For the same reason that a candle exists in these days of electric lights. Local conditions govern its use.

H. W. HARDINGE.

New York, February 1.

HIGH-DUTY GRAVITY STAMP-MILLS

By PETER N. NISSEN

(December 30, 1911)

*Although the gravity stamp-mill has been in use for generations, I venture to state that it is not as generally understood, and consequently not as thoroughly appreciated, as its merits warrant. Possibly this is due to the great simplicity and seeming crudeness of its construction and the tendency to regard anything simple as unimportant and unworthy of serious study. However, the most interesting facts are often concealed in the simplest forms, and it is safe to say that nothing is of too little importance to repay investigation. This is especially so with the gravity stamp-mill, for in spite of its simplicity and seeming crudeness, it is susceptible of the most delicate adjustment, its efficiency depending in a great measure upon the skilled attention devoted to it. It is interesting at this stage to find, as a result of reliable tests, that the well designed gravity stamp has proved to be the best machine for crushing, within the prescribed limits, as will be shown in these notes.

It has been demonstrated in many fields that in the items of power, labor, wear and tear, the cost of running stamp-mills is

*Excerpt from a contribution to the *Jour.* Chem., Met. & Min. Soc. S. A.

lower than for any of the other known crushing machines. In addition, the stamp-mill covers a wider field in its work than competitive machines, since the size of feed for stamp-mills ranges from over 3 in. to fine sand; therefore, its position as a crushing machine is unique. Every practical man knows that it is almost impossible to maintain the product from ore-breakers at the required size for more than a short period. This fact adversely affects work by rolls and Chilean mills, the feed for which must remain within the size governed by the diameter of the rolls or edge runners if they are to perform efficient work. Tube-mills are limited in their feed to a size not exceeding 3/8 in., while ball-mills, although not so limited to size of feed, cannot possibly compete with stamp-mills in view of the enormous wear and tear on balls and liners in crushing the ore of the Witwatersrand.

The history and development of this excellent machine has been ably recorded by many others; it is therefore not my intention to attempt any further elucidation of the subject, so far as the development of the multiple stamp-mortar battery, having units of five stamps, is concerned. It is generally agreed that, in spite of the length of time this machine has been in use, and while many improvements have been made in an effort to get the best results, it still remains a very imperfect machine. Much thought has been given to the proper design for the rectangular mortar for five stamps for different conditions of service, and many types have been produced, all of which adhert to one general principle and differ only in detail. The principle of having five stamps in one mortar has rarely been assailed, but it rather seems to have been taken for granted that it could not be improved upon. That this has been the case with such an important and useful machine, must always remain a surprising fact to the thoughtful observer.

Since the function of the stamp is one of crushing the greatest quantity of ore for a given expenditure of power, it follows that to get the highest efficiency from each blow of every stamp, the conditions of feed are most important. Regardless of the order of drop, and in spite of the different theories advanced, it is evident that all the stamps in a multiple-stamp mortar are not doing the same amount of useful work; it must be recognized that the uncrushed ore in the mortar is not distributed evenly over the dies, so that the coarser pieces, and therefore the pieces first struck, are not always arranged near the centre axis of the falling body. This lack of control of the feed is, in my opinion, largely responsible for broken shoe necks, broken stems, worn guides, and uneven wear of dies. The uncertainty of having the uncrushed ore, particularly the large pieces, in the centre of the die face, renders it impossible to expect really efficient work from the stamp. Of equal importance with a controlled feed is the positive discharge of the crushed ore when reduced to the desired size. The rectangular mortar, having a screen on one of its longer sides, does not suggest a logical method of dealing with the problem. By the order of drop necessary to force the ore from one die to another, waves of motion are

set up from end to end of the mortar, and only that material adjacent to the screen can be discharged. Double discharge mortars might have improved these conditions, were they not still more difficult to feed, owing to increased width and the necessity of having to introduce the ore higher in the box, besides other faults of design.

It is not within the limits of commercial possibility to make the multiple-stamp mortar of sufficient weight in itself to secure the necessary inertia for effective crushing. It is therefore important that it should be securely fastened to a foundation having great weight and solidity. In a mortar with the blows of the several stamps acting on different sections of the casting with great rapidity, the greater the length the more difficult it is to hold securely on the foundation. Regardless of the skill exercised in fitting the mortar to the foundation, a certain rocking tendency exists, due to its being struck at alternate ends by the falling stamp. It is difficult to keep the foundation bolts tight; in any case, they are liable to stretch and finally break, due to the strain. Certain internal stresses exist in the mortar casting, due to its design, and this eventually breaks by the repeated blows.

Coming to stamps weighing about 1250 lb. and up to 2000 lb. each, we find the same diameter shoes and dies are used in all cases, so that the mortar need be no longer. Attention has been drawn to this fact by several writers, and rightly so, for there must be a correct relation between the area of the crushing members and the weight of the stamp, on rock of a certain hardness and friability, to get the highest efficiency from the blow. Given the same length of mortar, to increase the weight of the stamp, it becomes necessary to use longer heads and heavier tappets. It is apparent that this is not the most desirable way of obtaining the object in view, because it raises the centre of gravity and increases the distance between the face of the die and the lower guide. A more rational way of accomplishing this object would be to increase the diameter of the head. This, however, would demand a longer and heavier mortar to accommodate the larger stamps and thus exaggerate the rocking tendency, besides increasing the distance between the cam-shaft bearings, which is undesirable. It is well known that there are limitations to the diameter of cam-shafts, since the larger the shaft the greater the velocity of the cam at the point of engaging the tappet, due to the greater distance between centre of cam-shaft and centre of stem.

It has been proved that heavy stamps are better crushing machines than the lighter stamps, within certain limits; therefore, since the heavy stamp is desirable, and as ordinary five-stamp battery construction does not lend itself to use with the very heavy stamp, a new design had to be evolved. To render the use of the heavy stamp possible, different principles of design should be adopted, so as to harmonize the desired greater crushing capacity with sound mechanical construction.

HIGH-DUTY GRAVITY STAMPS

By H. Stadler

(February 17, 1912)

*The primitive structure of gravity stamps, their clumsy appearance, contrasted with a compact and sweet-running dynamo, has been a perpetual temptation to the mechanical engineer to replace them by some more ingenious machinery. While all new appliances invented for the purpose have so far ended on the scrap heap, even before the 'puffs' proclaiming 'the passing of the stamp' had made the rounds of the press, the stamp still shows a vigorous vitality readily explained by the following considerations:

1. The countless alternations of stresses, inevitably produced in breaking hard rock, loosens all the parts of complicated machinery in time, and only the toughest and bulkiest materials can stand the destructive effect of resilience in the long run.

Table I

Distribution of Gold contents over Grades in Battery pulps.

Grade.	Apert.	600 mesh. Orig. 5·4 Dwt.		Tube Mill. (·216in. × ·446in.) Orig. 3·9 Dwt.	
		Weights.	Ass. Val. per ton.	Weights.	Ass. Val. per ton.
	in.	%	Dwt.	%	Dwt.
+30	·0197	9·75	1·4	53·25	2·2
40	·0146	20·50	2·0	5·25	3·5
50	·010	8·50	2·8	7·75	3·8
+80	·006	17·0	5·1	7·25	7·6
−80	·006	44·25	8·1	26·50	6·3
		100%		100%	

2. The great adaptability of the stamp to varying working conditions and its range of duty, especially with regard to size of feed. It is helped in this direction by a most satisfactory automatic feeder, a 'thinking' mechanical device which regulates the feed according to the digestion of the stamp.

3. Its 'life' is practically unlimited, as all its parts can continuously be replaced by repairs made by mine workmen.

4. Crushing by impact, as done by stamps, is extremely well suited for freeing the metallic gold and pyrite from their gangue. The enrichment of the finer grades in gold content, as illustrated by the following figures, explains the great amenability of battery pulps to amalgamation.

In an appreciation of the undoubted merits of the Nissen stamp it is to be regretted that it has not been given a more thorough test.

*Author's revised copy of paper printed in *The South African Mining Journal,* December 2, 1911.

The summary of the 'exhaustive' tests carried out at the City Deep, Ltd., by the Central Mining & Investment Corporation, Ltd., consists of but a few picked tests made under working conditions, which vary just enough to allow a correct conclusion to be drawn as to the effect of these variations. My method of computing crushing efficiency, as practiced during the last three years in the experimental work carried out on behalf of the Mines Trials' Committee, has been proved to give highly satisfactory and reliable results, and it is therefore gratifying to me to have it confirmed by H. S. Ball,[1] who tells us that after careful examination "it was unanimously decided that this method would be applied to all crushing tests of the McGill University." However, here in Johannesburg, the Central Mining & Investment Corporation, Ltd., which contributes to the costs of the Mines Trials' Committee, and is largely represented in that body, seems to give a too liberal interpretation to the maxim of the Lord, not to let your right hand know what your left does. In order to make the data of the summary of Mr. Nissen's paper more legible, the gradings have been replaced by the one representative figure of their mechanical value, and the three factors determining the energy invested in the stamps by the one value of foot-pounds per second.

The mechanical values of the gradings are practically identical for both types of stamps and exactly correspond to those of the curve established for varying coarseness of battery mesh. This confirms the statement made in 1909[2] (referring to lighter stamps) that the gradings of screen pulps are not materially affected by varying working conditions of stamps, but are practically exclusively determined by the mesh aperture, and it shows that the principle also holds good for heavy stamps, at least within the range of the coarse meshes used in the above tests. The horse-power consumption has been calculated from the value of the foot-pounds per second, with an allowance of 25% for loss in friction (in an old mill 30% is nearer the truth). By this method the power consumption works out lower, in all tests, for the ordinary stamps, while the table of the Central Mining & Investment Corporation, Ltd., with one exception, gives lower values for the Nissen stamp. Since the mechanical arrangements for lifting the stamps are practically the same in both types, the horse-power (as theoretically established) must be taken as giving more correct values than those obtainable by unreliable direct measurements. A possible error made by taking exactly the same height of drop for both cases would be in favor of the Nissen steamp, which, being heavier, might have its lift a little increased in consequence of a greater compression of the rock-bed. Until satisfied that all necessary precautions have been taken for exact tonnage measurements, the duties given in the table must be accepted with reserve. In

[1]'The Economics of Tube Milling,' Bull. Inst. Min. & Met., August 30, 1911; *Mining and Scientific Press*, September 23, 1911.

[2]Mines Trials Committee, second report of Sub-Committee.

truth, they are in some cases not logically related to one another, nor to the conditions of the individual tests. The greatest drawbacks for such tonnage measurements are the continuous fluctuations in the proportion of 'fines' contained in the feed. From the following complete grading of a battery feed it will be noted that, while the mechanical value of the total feed is 0.424 E.U., or say

TABLE II

Example of full Grading of an Average Battery Feed

I.M.M. Mesh.	Mech. Value. of Grade.	Weights.	Mech. Value.	Gradings reduced. to +1in., and —1in. portion. Weights.	Mech. Value.
	E.U	%	E.U	%	E.U.
—	4·73	27·97	1·323	41·5	1·963
—	2·90	13·15	0·381	19·3	0·560
—	2·16	7·93	0·171	11·8	0·255
—	1·56	5·22	0·081	7·8	0·122
—	1·08	3·76	0·040	5·6	0·060
—	0·50	5·01	0·025	7·5	0·038
Apert.	0·08	4·17	0·003	6·5	0·005
+ 1in.	0	67·21	−2·024	100%	−3·003
—	1	7·10	0·071	21·7	0·217
¾ in.	2	—	—	—	—
—	3	7·93	0·238	24·2	0·726
½ in.	4	—	—	—	—
—	5	3·55	0·178	10·9	0·545
¼ in.	6	—	—	—	—
—	7	3·34	0·234	10·2	0·714
⅛ in.	8	—	—	—	—
holes	9	2·33	0·210	7·2	0·648
5	10	—	—	—	—
—	11	1·90	0·209	5·9	0·549
8	12	—	—	—	—
—	13	0·81	0·105	2·5	0·325
12	14	—	—	—	—
—	15	0·91	0·137	2·8	0·420
20	16	—	—	—	—
—	17	0·77	0·131	2·4	0·408
30	18	—	—	—	—
—	19	0·92	0·175	2·9	0·551
50	20	—	—	—	—
—	21	0·97	0·204	3·0	0·610
+ 80	22	—	—	—	—
− 80	27	2·06	0·556	6·3	1·701
—	—	100%	+2·448	100%	+7·514
Less Mech. Val. of + 1in. Grades			2·024		
Mech. Value of total feed			0·424		

0, the + inch portion (about $^2/_3$), taken alone, works out at — 3.0 E.U. (the — indicating coarser than the one-inch, or 0 — grade), and the — inch portion at + 7.5 E.U. (Table II).

Assuming as an extreme case that temporarily only the + one inch portion (about $^2/_3$ of the total feed) goes to the mill, the work done by the stamps must be credited with three additional energy units (E.U.). On the other hand, if the — inch portion (about $^1/_3$)

is considered alone, 7.5 E.U. should be deducted from the mechanical value of the screen pulp as work already done by the preliminary crushers. Stamps dealing with such a fine feed necessarily, therefore, have a correspondingly increased duty. In the Giesecke mill tests, where exact tonnage and power measurements were possible, it was ascertained that the variations of the percentage of 'fines' easily affected the duty by about 2 tons per hour, or 15% of the duty. Without special arrangements and precautions the obtaining of representatives average samples of a day's run is practically impossible, and for comparative tests it is just as accurate—or inaccurate, if you like—to assume the feed to be uniform and of the 0 grade.

As stamps under exactly similar conditions do invariably the same class of work, it is more advisable to extend experimental tests over comparatively short periods only, during which the working conditions can be perfectly controlled and maintained. By making tests in one series for each variable, the use of the regular curve obtained is not only a valuable check, but also the only way to arrive at definite conclusions as to the effect of the variables on crushing efficiency. Allowing for all these uncertainties in the data given, the fact remains that the Nissen stamp shows a decidedly higher efficiency than ordinary stamps.

In contrast to life, which is created by life, but doomed to pass away into spheres hitherto inaccessible to human research, matter and energy can neither be produced nor destroyed, but only converted. Theoretically, no efficient machines exist, and what we call in practice efficiency is only the relative amount of useful work done with regard to the special purpose for which the machine has been designed. With this reserve the greater effiicency of the Nissen stamp, or the relative loss of energy in the old stamps, may be explained as follows:

1. The roundish shape of the particles leaving a 5-stamp battery proves that a certain amount of the reduction work is done by attrition and abrasion during the shifting and knocking about of the particles in the mortar box. In the one-unit Nissen stamp the particles are less subject to this action and less energy is lost by useless water churning.

2. The radial arrangement of the screen allows an immediate rejection of the already finished product, leaving behind a clean material, which, since an ample percentage of its apparent volume is free interstices, is not subject to cushioning.

3. The accumulated energy at the moment of the impact of a falling stamp is not only distributed over the surface area of the shoe, but it is also transmitted in depth. With a too thick or uneven rock bed, the amount of energy absorbed in packing the material, before crushing by compression can take place, may be so great that the crushing work done by direct impact becomes almost nil. The maintenance of an even surface of shoes and dies and the

TABLE III

	Tyler T. C. 9 Mesh.				9 Mesh.		⅜ in. sq. Mesh.			
	0.205 by 0.536 in.				0.277 in. sq.		0.375 in.			
	Nissen Stamp.	City Deep.	Nissen Stamp.	City Deep.	Nissen Stamp.	City Deep.	Nissen Stamp.	City Deep.	Nissen Stamp.	City Deep.
Ft. lb. sec. per stamp	23.49	21.99	23.43	21.94	24.23	20.95	28.11	22.54	24.21	20.92
*Power consumption per stamp, hp.	5.34	5.00	5.33	4.98	5.51	4.76	6.39	5.12	5.50	4.75
Duty per 24 hr. per stamp, tons	24.47	18.26	23.73	19.90	29.81	20.95	36.69	22.72	27.74	24.34
Duty (theoretical)	(24.50)	(23.00)	(24.50)	(23.00)	(24.50)	(21.00)	....	(24.00)	(26.00)	(22.50)
Mech. value of pulp, E.U.	17.25	17.44	17.31	17.52	16.97	16.88	16.88	16.57	16.28	16.31
†Relat, mech. efficiency per hp	79.00	63.69	90.00	70.00	91.80	74.30	108.40	73.50	111.70	83.60
Increase in efficiency	24.4%		28.6%		23.5%		47.5%		33.6%	

$$* \frac{\text{Ft lb. sec.} + 25\%}{550} = \text{hp.} \qquad \dagger\text{Rel. mech. effic.} = \frac{\text{Duty (tons per 24 hr) by mech. value}}{\text{hp.}}$$

facility for weighing the shoes as they wear, are far-reaching factors of inefficiency hitherto beyond our control.

4. The blows of a one—unit Nissen stamp are bound to be harder and shorter, as the mass of the anvil formed by the thick and heavy bottom of the mortar box, placed in the centre of the foundation, is certainly more solid and rigid than in the ordinary 5-stamp mortar boxes, where rocking may take place. The vibrations due to the shocks being shorter, less energy is lost in shaking the building and surroundings.

The lower efficiency shown by the ordinary stamps is partly due to energy lost in deformation within the limits of elasticity, or permanent deformation before reaching the breaking point. To some extent, however, the loss of efficiency is only apparent, as the lower duty is to a certain extent balanced by a larger amount of crushing work performed in the very finest grades as a consequence of the attrition and abrasive action. By taking for the mechanical value of the —200 grade a standard value of 28 E.U., the work done in producing ultra fine slime is not accounted for, but this is quite justified, as, for Rand ore, grinding finer than to 200 mesh is mere waste of energy. From these premises may be made the deduction that the superiority of the Nissen stamp would be still more pronounced when crushing through finer mesh. The policy of ultra fine grinding by wholesale tube-milling, just now in fashion in the metallurgical department controlling the Rand Mines, has been, unfortunately, no inducement to them to extend their 'exhaustive' tests in this direction, and the Nissen stamp has therefore not been given the chance of demonstrating its great possibilities as a fine grinder.

Hearty thanks are due to P. N. Nissen for his able and interesting paper, which, I think, leaves on all the impression that this stamp marks a great and real progress in stamp-milling. The new departures are based on sound and theoretically correct principles, and the details of the mechanical arrangements are practical and handy. This is confirmed by the best judges of new inventions in this connection—by the shiftmen, the men who have to work with it. The calculations of comparative working costs and capital expenditure may be open to criticism, but, compared with the other great advantages claimed and proved, the differences are too small to justify protracted squabbling about the matter.

STAMP-MILL PRACTICE AT THE HOMESTAKE

(January 11, 1913)

The table below shows the weight, dimensions and approximate life of stamp-mill parts in the mills of the Homestake Gold Mining Co., as given in the recent paper by A. J. Clark and W. J. Sharwood on metallurgical practice at the Homestake.

	Material.*	Dimensions.	Approximate Weight.	Approximate Life.‡
			lb.	
Cam-shaft ..	Wrought iron ..	5·36 in. diam.	1050 to 1085 †	4 years.
Cam	Chrome steel ..	2·5 in. face × 35 in., hub 12 in. diam. × 5·75 in.	262	No data.
	Cast iron ..	" " ..	256	2 years and up.
Tappet	Chrome steel ..	9 in. × 12 in. ..	150	No data.
	Cast iron ..	.. " ..	132	20 months and up.
Stem .	Wrought iron ..	3·125 in. diam. × 15 ft.	380 to 390	4 mths. between breakages.
Boss-head..	Cast iron .	9 in. diam. × 18 in.	236	4 to 12 years; average 6.
Shoe ..	Special chilled cast iron	9 in. diam. × 8 in.	145	60 to 90 days.
Die.. .	Hard cast iron..	9 in. diam. × 5·5 in.	110	30 to 35 days.
Mortar ..	Cast iron ..	"	5500	8 years.
Screens ..	Cold-rolled re-annealed O.H. steel	9 in. × 50 in. × 0·035 in.	4	10 to 16 days.
Plate ..	Lake copper ..	4·5 ft. × 12 ft. × 0·125 in.	320	

* Iron castings, with the exception of shoes, which are purchased in lots of several thousand, are cast in the Homestake Foundry, and charged to the various departments at a uniform rate of about 3 c. per lb.

† The shafts in different mills differ slightly in length.

‡ These figures are averaged from a record covering breakages over the whole plant for about three years. At the North-side mills, with lower and slightly faster drop, and somewhat softer ore, the average life of mill parts is considerably longer than in the Lead mills. In the case of chrome steel cams and tappets the period covered by the record is too short to give even approximate data.

BALL-MILL PRACTICE AT KALGOORLIE

By M. W. von Bernewitz

(July 15, 1911)

So much has been written about the modern stamp-mill that other efficient pulverizers are apt to be overlooked; therefore I make no apologies in presenting this discussion of the modern ball-mill, in this case the Krupp, in the hope that it may be of use to others who have plants in operation, or contemplate their erection. It is doubtful whether the dry-crushing ball-mill is operated with greater efficiency and less cost in any other mining centre in the world than in Kalgoorlie. We have had great experience with them, and with a constant interchange of ideas, each plant is well abreast of its neighbors, until there is practically no difference of opinion as to operation.

The ores in the 'Golden Mile' do not vary much in composition, and, taken on the average, they may be said to be hard to crush. The ore in most of the mines averages 60% silica, the Great Boulder showing as high as 75%. Summary descriptions of the ore milled in the dry-crushing plants follow: the Associated has little schist, rather above average in silica; hard and highly mineralized. The Associated Northern main lode is of schistose character; west

lode hard and silicious. The Chaffers ore varies from schistose to hard flinty quartz, with 3 to 5% pyrite. The Great Boulder has very hard cherty quartz with a little schist. The Kalgurli ore is quartz-diabase, varying in hardness according to depth, from schistose to highly silicious; average tough. At the Perseverance the ore is schistose in structure, and contains pyrite and magnetite throughout. South Kalgurli ore averages 60% silica, with little schist, and is considered hard.

The following analyses are from the Associated Northern and Perseverance mines:

	Per cent.	Per cent.
Insoluble	60.60	63.46
Aluminum oxide	5.75	2.06
Iron bisulphide	5.32	6.80
Carbonate of lime	11.19	13.42
Carbonate of magnesia	5.29	6.93
Carbonate of iron	10.23	6.80
Alkalies, undetermined	1.62	0.53
	100.00	100.00

The average ore from the mines is dry enough for milling, although the driest holds from 0.5 to 1% of moisture. Anything above this amount retards crushing. In a stamp-battery, or other type of crushing machine, a great deal of the efficiency depends on having massive foundations. So it is with the ball-mill. A mill on a solid block of concrete will crush better than one on a timber or steel girder, no matter how well braced. A springy support is bad for the gear, and does not allow the mill to be stiff, as it were. This has been noticed especially in the Associated plant built eleven years ago, in which 10 mills were resting on timber girders, and two on concrete piers, the latter doing far better work than any other pair in the plant. The former foundation consisted of a masonry bed, on which stood 12 by 6-in. steel girders with brackets riveted on to support two longitudinal pieces of jarrah 12 by 14 in., across which were bolted three pieces of 12 by 14 for each mill. This would appear to be stiff enough, but the vibration was still bad. Now the big mills are on concrete foundations, reinforced with old wire rope and rails. The Associated Northern, Chaffers, Great Boulder, and Perseverance ball-mills are set on massive concrete foundations; the last mentioned is especially good. Those at the Kalgurli are set on steel girders 8 by 6 in., well braced and apparently firm; while in the South Kalgurli plant they are bolted to a frame of 12 by 12-in. timbers, well braced and standing on a concrete foundation. This seems a small structure, but is free from vibration.

As soon as the foundations for a ball-mill are sufficiently set, and the four bearings for the mill-shaft and pully-shaft are lined up, the building of the mill can be done rapidly; and with eight men who understand the work a mill may be erected in 24 hours. Everything is simple; the main walls are bolted or riveted to the cast-iron naves, and when in their proper places, are keyed to the

mill-shaft. The side-liners are bolted loosely until all the mantle-plates with the grinding plates are in position, then everything may be tightened, all bolts having check-nuts, excepting those on the mantles holdings on the grinding-plates, which are riveted. The fitting of the scoops, baffles, and inside and outside screens is quite a simple matter. The feed-hopper is in two pieces for bolting around the shaft, and is lined for the wear of ore dropping in from the feeder. Unless motor-driven, each mill has a fast-and-loose pulley on the pinion-shaft. There are two sizes of mills at work in Kalgoorlie, with the following dimensions:

	No. 5. Inches.	No. 8. Inches.
Diameter	89	106
Width	46	54
Mantles for plates	10	12
Inside screens	5	6
Outside screens	10	12
Area of outside screens each	46x26	54x24

The latter item is the full area, not deducting that part of the screen covered by the wooden frame. Approximately, the net screening surface on each mill is 70 and 100 sq. ft. respectively.

The mantles carry four or five grinding, and one perforated plate, which are tightly bolted together. The bolts have a feather on the tapered portion which fits into the plates, thus preventing turning during tightening. Bolted to the bottom end of the mantles are the baffle and scoop plates, which return the oversize to the mill. The whole mill is enclosed in a tight-fitting casing, which is made in convenient sections for removal during repairs. The spur-wheel is keyed to the mill shaft, is usually of cast iron, and should last, if well greased, fully four years, while the pinion lasts about half that time. The loose pulley runs on a sleeve, which must be well greased to prevent seizing. At one of the new mills at the Associated, a new idea for loose pulleys has been devised, a cast-iron bracket with an extended hollow sleeve, bolted to the plumber block farthest from the mill. The pinion shaft revolves in this sleeve, and the loose pulley rests on it. When the mill is working, the pulley is at rest, thus preventing much wear at this point.

It is found that a good, stiff, light grease is best for all bearings on a ball-mill. A few turns of a grease cup twice in eight hours is generally sufficient. For the gear, old, though clean, grease from other bearings is good enough. Properly fitted gearings, when well greased, make little noise. When driven from a main driving shaft, a tight and a loose pulley are required; but when motor-driven, the former only is necessary. The Great Boulder people are fitting clutch pulleys on the main shaft with satisfactory results. an 8-in., 6-ply balata or rubber belt is strong enough for a No. 5 mill, and a 10-in., 8-ply for a No. 8. From 24 to 26 r.p.m. is the correct speed for the No. 5, and from 21 to 24 r.p.m. for the No. 8. It is generally admitted that the latter speed for the larger mill is

quite safe and shows much greater efficiency, although the makers stipulate the lower speed. Mills are loaded with one to two tons, of 5-in. steel balls, according to size and hardness of the ore to be crushed; there is no advantage in using several sizes of balls. Indicator diagrams at the Associated Northern and Associated have shown that the No. 5 mill takes from 18 to 23 hp. when in an average working condition, and the No. 8 consumes 60 hp. Where motor-driven, as at the Perseverance and South Kalgurli, the ammeters register 60 to 65 amp. at 550 volts.

It is found advisable to have the storage-bin for coarse ore a few feet behind the ball-mills, as it allows the construction of a proper chute with a gate, and a long feeder, thus giving a steady flow of ore. When the bin is right over the mills, as at the Associated as originally erected, this cannot be relied on, ore often rushing out and filling up the mills. For the protection of employees, keeping the plant free of dust, and the better working of a ball-mill, a strong fan attached to the mill by large piping, is very necessary; and a plant properly arranged in this detail is a pleasure to work in. A couple of Sturtevant or similar type fans, about 4 ft. diam., running at 800 to 1000 r.p.m., should keep eight No. 8 mills clear; and a 3-ft. fan like that at the Associated Northern, running at 800 r.p.m. is very effective for three No. 5 mills. A 4-ft. motor-driven fan at the Associated takes 5 amp. The suction and delivery pipes should be fitted with sliding doors to facilitate cleaning. The draft in the suction-pipe at the Associated Northern averages 0.33 in. The fan generally blows the dust into a canvas-lined house, in which it readily collects, or as at the last-named mill, it is blown into the second hearth of the Merton furnaces. At the Perseverance, the dust is collected in cyclone arresters, thence into the main furnace flue. A strong drift aids in keeping the screens clear, especially when the ore is at all damp, as the steamy air is drawn away. When dry crushing with stamps at Waihi in New Zealand, in 1894 to 1899, the beneficial effect of a fan was very noticeable. The amount of dust lost in dry crushing is difficult to estimate; but from observations I should say it would total 1 per cent.

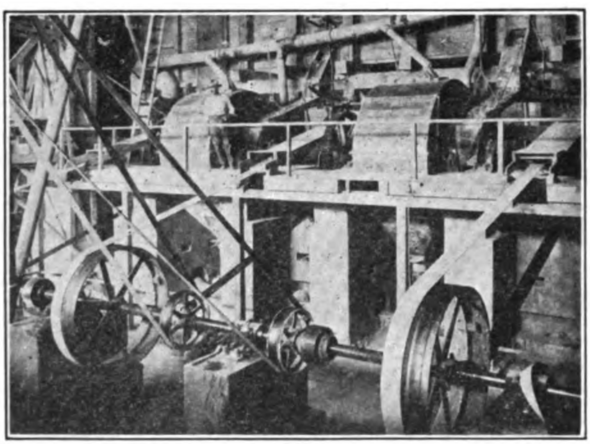
The ball-mill feeders are usually of the grasshopper or shaking-tray type, that is, a sloping tray with sides of 6 in. high at an angle of about 25°, suspended at the discharge end of a chain or rod from a beam above, and the top end supported by two iron legs, fitted loosely to the under side of the tray, and to a timber support near the floor. At the back of the feeder tray is a tappet, and on a shaft is a single, double, or three-armed cam, adjustable to give up to 1 in. lift to the feeder. This pushes the feeder back, which, when released, drops forward by its own weight, discharging an even feed into the mill. It is the custom now for each mill to have its own feed-shaft driven by a 3-in. belt from a pulley fitted to an extension of the mill axle. The mills at the Perseverance are so fitted, the whole being very neat and effective. South Kalgurli mill feeders are arranged with a perforated tray, through which the fine ore passes on to a shaking tray fitted with a 27 by 27-in.

screen, this in turn screening all that portion already fine enough, it passing to the conveyor beneath. One was tried for some time at the Associated Northern, but it did not increase the capacity of the mill to any extent. The arrangement requires close watching. Feed naves are arranged for left and right-hand feeding; by this means mills may be run in pairs, the feed hoppers being reasonably close together, and it may be remarked here that mills should not be too close together, as plenty of space is required for repairs, etc. The balls are fed in either through the manhole door or feed-hopper, and ore fed in until the balls are well mixed with it. A mill should not be started without feed, as the balls knock the plates about badly. Unless a mill is motor-driven, it would be difficult to arrange for an automatic feeder. I have not heard of any, but think that one could be fitted up depending on the increase of power consumed over a set point. If a mill takes 60 amp. with a full load, and is overfed to 62 amp. then some automatic gear could shut off the feed. Like many other machines, the operation of a ball-mill is governed by the sound. This is an important point to impress on mill-men, as a poor attendant may easily reduce the efficiency of a mill 25% by inattention. With practice the operator of a ball-mill gets expert and can detect a light feed from some distance; a mill running empty simply roars; when light there is a clear metallic sound of balls striking one another; when just right, there is simply a rumbling noise; while in a mill that is full, the balls and ore are carried high enough to fall directly on the mill axle or shaft. This is very bad work, as the shaft is liable to be damaged and worn, and crushing is reduced to a minimum. The remedy for too much feed is, of course, shutting it off until the correct sound is heard. A mill-man not too sure about his feeding can easily see this by stopping the mill, and looking into it through the feed nave with a light. The mass of balls and ore should be evenly distributed, and no bare balls should be visible.

The operation of the Krupp ball-mill may be described as follows: The mill is a gear-driven, steel-lined drum, charged with one or two tons of steel balls, revolving at 24 to 26 r.p.m., according to size. The ore crushed by the balls, passes through perforated grinding plates, and is screened by the inside or coarse screen, the oversize being automatically returned to the interior of the mill; while what passes the inside screen is screened by the outside or fine-wire screen, the oversize being also returned to the mill for further grinding. The action of crushing and returning the oversize is ingenious, to say the least, and it will be be seen that as soon as the ore is crushed fine enough, it must pass out of the mill. Here lies the efficiency of the ball-mill. In a 5-stamp battery, there is a great deal of unnecessary pulverizing given the ore due to its not being discharged as soon as it is fine enough. In the ball-mill there is little of this, and any machine that discharges its product when it reaches the required size will show high efficiency. As regards the action of the balls, it has been stated that crushing is done by impact; but from careful observations I am perfectly satisfied that this is not so. The only impact which

might occur is when the balls drop from one grinding plate to another, a distance of 5 in. in a newly lined mill; and then only some very fine particles would be crushed by impact at this point. The action of the balls is one of pure rolling, grinding, and abrasion. They are lifted up about level with the mail shaft or axle, and then roll back, tumbling over one another, rubbing and grinding the ore fed in. Of course there is a great deal of attrition between the pieces of ore apart from the work of the balls.

This is not mere supposition, but fact, as I have myself seen the action. At the Associated mill, four No. 5 mills are crushing oxidized ore which carries 5% moisture, so coarse screens are used; in fact, the wire screens were removed and the inside ones only used; these being punched steel with 30 holes per square inch. In crushing this ore no dust is made, so by the aid of a portable electric lamp, placed well down in the feed hopper, the balls and ore could



NO. 5 MILLS AT THE ASSOCIATED NORTHERN

be seen plainly. The action observed was simply that the balls and ore were carried up nearly level with the shaft, and on reaching this point began to roll back. There was 1200 lb. weight of light-weight balls used on soft ore, rolling and tumbling about and grinding and rubbing the ore between them. It was a most interesting sight, fully proving that there is no impact grinding. To further prove this, one has only to observe the ore in a mill when it is stopped for repairs. Down to the very small particles, say 6-mesh, it is smooth and round, just as if water-worn in a river where the stones are rounded from a rolling action. Pieces of steel, which come in from the mine at times, are rounded in the mill, although they hinder good work to some extent. If impact were the action in a ball-mill, dynamite, which also finds its way in with the ore, would explode, as it does in a Griffin mill; but it does not, and I assume that it is simply rubbed away. F. B. Allen (director of

technical education in this State, with G. Larcombe, instructor in geology, have kindly examined, under the microscope, some ball-mill product which was wet-screened to clean the particles from dust, to see whether they had a sharp fracture or not, with the result that they show the + 30 screen product to be somewhat rounded, and the + 120 to be with sharp, ragged edges. This shows that the final particles are split by the rolling of the heavy balls, not necessarily by the drop of a ball, as in a tube-mill, as described by H. Fischer in several of the leading technical journals in 1904.

Table I

Name	Number of Mills	Weight of Balls, lbs.	Screens	Speed, r. p. m.	Horse Power	Drive	Capacity Daily, tons	Steel Consumption, lb. per ton	Life of Grinding Plates, days
Associated	4 No. 8	4480	30x30	21	60	Belt from shaft	92–95	0.50	170
	6 " 5	2240	25x25	25	23		43		
Associated Northern	3 " 5	2350	27x27	26	18	Belt from shaft	40	0.32	270
Chaffers	3 " 5	2300	27x27	25	25	Belt from shaft	40	0.74	180
Great Boulder	4 " 8	4480	30x30	24	60	Belt from shaft and friction clutch	80–90	0.64	105
Kalgurli	9 " 5	2200	26x26	25	25	Belt from shaft	40	0.45	300
Perseverance	8 " 8	4400	27x27	24	60	Motor and belt	100	0.65	118
South Kalgurli	3 " 8	4480	27x27	24	65	Motor and belt	95–100	0.47	210
	1 " 5	2800			30				

In Table I will be found the consumption of steel and the life of liners in a mill. Steel consumption is gauged by the weight of balls consumed per ton crushed, and not only by the total balls and liners used, as in a battery where shoes and dies are calculated. The average steel consumption in the ball-mill plants here is 0.55 lb. per ton, while the hunch or grinding plates last about 190 days, perforated plates the same time, and side liners about double this life. Manganese steel of the Hadfield or Krupp make is very satisfactory. As regards the capacity of ball-mills or other machines, the efficiency depends upon the quantity crushed per horse-power consumed. A No. 5 Krupp mill crushes 43 tons daily of ordinary sulphide ore, and the No. 8 100 tons; dividing this by the horse-power consumed, 18 to 23, and 60 to 65, 2.1 and 1.6 tons per horse-power respectively is obtained. In a small plant of 100 tons daily capacity, the No. 5 is a useful unit, but in a large plant the No. 8 is the best size. They have large capacity, run smoothly, and occupy little more space than the smaller unit. The quantity of ore crushed as given above is from actual weighing in some cases, and from the tonnage crushed divided by the hours run in others. On quartz the capacity of a No. 5 mill is lower, being about 36 tons per diem. Excellent opportunity of finding out the capacity was had at the Associated Northern, as all the custom ore is weighed.

It is impossible to prevent damp ore of from 1 to 3% moisture, coming into the plant. Then the ball-mill efficiency falls off greatly and the wear and tear is heavier; the feed must be light to prevent the screens clogging, as a steaming is set up, making everything sticky. When crushing dry ore, the inside of the mill gets very hot from friction, a recent test showing up to 135°F. when the outside temperature was 97°. The Kalgurli used to dry the damp ore before milling, but rotary drives are a nuisance. The soft oxidized ore from the mines is easily crushed in ball-mills, as at the Associated and Associated Northern, by using only 1200 lb. of balls, taking off the outside screens by passing it through the inside or coarse screens. Pans reduce this product quite easily. Actual tests show that a No. 5 mill will crush 82 tons per day, but for 5% moisture the capacity would be greater. The Chaffers has one mill on this class of ore, wet-crushing 93 to 100 tons through 26 by 26-in. screens per diem, using 2400 lb. of balls.

It is advisable to open mills for inspection once a week, unless something goes wrong in the meantime, and every two months the balls should be taken out, weighed, and all balls under 1½ in. diam. discarded, as they are of little use in crushing, and the original weight made with new balls of 18 lb. each. Foreign matter, such as drill ends, bolts, nuts, spanners, etc., accumulate in mills, and as they hinder the smooth work of the balls they must be thrown out. One man at 11s. 8d. per shift of 8 hours can look after eight No. 8 mills; but in smaller plants the mill-man attends to conveyors, elevators, dust pipes, etc. Ball-mills should be fed with no larger ore than that which will pass a 3-in. ring. With fire ore the balls are apt to bed, as it were. Actual weighing has shown that a 3-in. feed is crushed faster and the wear of steel is less than when the feed is 1 in. with a quantity of fine.

Following are grading analyses from ball-mills in our different plants:

TABLE II

NAME	+30	+40	+60	+80	+100	+120	+150	—150
Associated	....	16.16	15.66	10.46	6.43	3.76	3.46	44.00
Associated Northern	....	11.20	20.10	8.30	11.80	2.20	2.00	43.60
Chaffers	....	10.25	13.19	8.60	7.25	2.70	1.71	56.30
Great Boulder	1.00	5.80	14.20	8.70	9.00	2.30	7.00	52.00
Kalgurli	....	8.20	18.00	13.40	9.10	2.40	2.20	45.30
Perseverance	0.20	3.80	20.60	7.80	5.40	4.00	3.80	54.20
South Kalgurli	....	10.20	21.20	6.00	2.60	3.10	1.00	55.90

With the exception of the Associated and Great Boulder, where the screens are 30 by 30, 27 by 27 is the size used. They are fairly uniform, although the — 150 is low at the Associated, Associated Northern and Kalgurli for some reason not understood. The ball-mill does not make so much — 150 as the Griffin and other pulverizers. The coarse screens used to be fixed to the mill by studs, but a local improvement was introduced in fitting them on

with 1½-in. angle iron, bent into a circle, and bolted. A large number of bolts are used in a mill, and their cost is high. For some reason, perhaps because of the expansion of the plates by heat, or loose plates, the bolts break and fall into the mill. Coarse ore then passes through the bolt-hole into the space between the mantle and screens, causing the mill to run out of balance. Experienced mill-men soon detect this. On account of this trouble, there is a difference of opinion as to the use of the baffle plates that are fixed to the return scoop plates in the mill.

The fine screens should be brushed once daily, if the ore is dry, and oftener if damp, with a wire brush. They last from 30 to 40 days, providing no stones escape and tear them. When the grinding plates get thin and nearly worn to the mantle plates, the mill requires re-lining. This is done in different ways; some mill-men prefer to have a spare set of mantles which are fixed to the plates at leisure. When the time comes for re-lining, these worn-out plates are taken out and the prepared plates fitted in; this requires about ten hours with six men. Others consider it best to leave the mantles in place, cutting off the old grinding plates and fitting on new ones as they proceed. This can be done in about the same time, but is awkward, heavy, and hot work inside the mill. There are several little points in ball-mill practice on which agreement will never be reached, but their effect is trifling. A No. 5 ball-mill, including foundations, may be erected for £600, and a No. 8 for £1000, bin and conveyors not included. It will usually cost about £300 per year for upkeep of a No. 5.

I have seen No. 4 size mills used in Broken Hill plants for wet-crushing jig middling that passed an 8-mm. screen; a No. 5 on the Lake View Consols crushing ore (sulphide), and the No. 5 at the Chaffers crushing the soft schistose oxidized ore. In the first instance I do not think the mills show great efficiency on such feed; in the second, the mill never had a real trial, as the grinding pans and tube-mills were already overtaxed; while at the last the ore is so soft that any comparison with stamps on hard ore cannot be used. Some years ago the Eclipse mine had ten 900-lb. stamps crushing ore similiar to that at the Chaffers at the rate of 10.5 tons per stamp per day through a 20-mesh screen; so the capacity of the Chaffers ball-mill of 93 — 100 tons through a 26 mesh is somewhat better than the 10 stamps. The wet mill product is:

Size.	Per cent.
+ 40	10.60
+ 80	16.12
+ 100	4.44
+ 120	3.56
+ 150	7.59
— 150	57.69

The ore is crushed in weak cyanide solution in the proportion of 1.6 to 1. The wear and tear in the wet mill is almost twice as heavy as in the dry, the grinding plates lasting 105 days.

Max Drott made some comparative tests on hard quartzite ores between a 5-stamp mill, and a ball-mill for the Krupp firm. The latter showed an output 40% greater than the battery, even the dry mill being better on the same ore. I am convinced that on a thorough trial, other things being equal, the ball-mill will show a greater output and efficiency than the stamp battery. Perhaps some one will take up this question and show that the modern heavy gravitation stamp is not the most economical crusher for its present work and that there are other machines that can do the work better. Some costs of ball-milling follow:

TABLE III

NAME	PENCE PER TON						
	Wages	Power	Spares	Stores	Repairs	Sundry	Total
Associated	6.45	13.78	6.73	2.67	0.49	......	30.11
Associated Northern	5.87	12.08	4.73	1.42	0.10	......	24.22
Chaffers	2.09	10.64	......	0.25	6.88	1.71	21.57
Great Boulder	1.81	9.75	......	6.26	......	......	17.82
Kalgurli	......	15.23	......	7.49	......	......	22.72
Perseverance	3.21	19.18	3.32	0.46	0.11	0.54	26.84
South Kalgurli	2.05	20.82	5.54	......	3.68	0.21	32.30

The methods of segregation of costs are not uniform, as will be noted at once. On some mines, spares, stores, and repairs are lumped together. Power costs are high on the Perseverance and South Kalgurli on account of the use of motor-driving. The Associated is in a stage of transition from the old No. 5 mills to the cheaper run No. 8 size. The Great Boulder has lately installed a high-class steam engine, and it is expected to reduce power costs to the above figure. In Table III above are given interesting details of ball-mill dry-crushing practice at Kalgoorlie.

HUNTINGTON MILL PRACTICE AT KALGOORLIE

By M. W. VON BERNEWITZ

(November 16, 1912)

It is doubtful whether a better insight into the working of the Huntington mill can anywhere be obtained than on the Kalgoorlie field, where at the present time there are ten mills at work. In the outside districts there are ten more, and at Ora Banda a large plant with four mills has just started. While this article deals chiefly with Huntington mill practice, an attempt has been made to discuss the treatment of low-grade oxidized ores at the small mines and a comparison is made with stamps on similar ores around Kalgoorlie, more especially at the north end, about three miles from the 'Golden Mile.'

To give an idea of the ores handled at these plants, a quotation from Larcombe's 'Geology of Kalgoorlie' will suffice: "In ores from the zone of oxidation much iron oxide is present, varying from ochreous varieties to hard compact ironstone, with which is associated much ferruginous clay or kaolin, resulting from the decomposition of the country rock. Quartz is also present, representing the silica remaining after the breaking up of the small veins characteristic of the deeper zones." The ore deposits of the north end are somewhat different from those at the south end of the field, however, but no doubt the oxidized product is similar, with the exception of containing less quartz. The veins at the north end are generally small, and in consequence much clay and ironstone can be expected in the oxidized product. An analysis of this ore shows:

	%		%
SiO_2	50 to 65	Al_2O_3	10
$CaCO_3$	7	Fe_2O_3	12
$MgCO_3$	5		

In these small mines the workings are by open-cut, or from depths of 150 ft. The Golden Dream, for example, in its last half yearly report, gives the width and average assays of three veins; 30 ft. of $1.80 ore, 15 ft. of $2.16 ore, and 30 ft. of $2.60 ore, 7000 tons worth $2.60 per ton being ready to mine. At several of these small mines the ore is hoisted by means of Taylor friction hoists. This machine consists of a single drum driven by a grooved or friction pinion, in turn driven by a motor. The man on the surface, by means of a long lever of 1-in. pipe, controls the hoisting of ore, or lowering the bucket or cage, as the case may be; therefore the expense of a hoisting engineer is eliminated. The ore is trammed to the mill. In some cases a crusher is used, while at others the ore is broken by hand to the size for milling.

For driving the machinery, either gas-engines or electric motors are used, costing 2 cents per brake-horse-power-hour for the former, and up to 5c. per unit for electricity. The Huntington mills are fed by Nelson (improved Challenge) feeders, belt or eccentric driven from mill shafting. The pulp flows over one long, or several short copper plates, and then to either spitzkasten or collecting vats, the slime going to dams. The clean sand is then shoveled into vats for a seven-day treatment with cyanide solution. The plants are fairly well designed, and certainly costs show much technical skill. Such mines cannot afford to have idle directors, and large staffs. Everybody works hard, and the owners are able to doctor their engines and motors, repair mills, cyanide the sand, clean up, and generally get satisfactory results. Every man I met on several rounds of inspection was always busy, but ever ready to impart any data needed, if available. At such mines the owners do not deem it necessary to go into exact refinements of tonnage, grading analyses, and costs. Those cited here are as given me, though many lack full details. The earliest Huntington mill plants on the Kalgoorlie field were

erected on the Associated, Oroya, and Hannan's Proprietary mines, which had three mills each. They stopped work in the order given, the latter working till last year, having a run of about 14 years, mostly on customs ore. When I came to Kalgoorlie, toward the end of 1899, the Oroya plant was crushing 100 tons daily of oxidized ore from an open-cut. This only averaged $6 per ton, and the company paid a dividend on such low-grade ore, at the time of high returns from the large mines.

The Huntington mill, generally of the 5-ft. size, is an excellent machine for crushing and amalgamating all soft ores, similar to those described, or those of a fairly quartzose character; and probably no machine will beat it at this work. Crushing is done by pressure largely, and impact slightly; say pressure-impact. A certain amount of crushing is also done by grinding, as the rollers must exert some rubbing action while pressing against the die-ring, revolving at the same time. On this class of ore, fully 50% is slimed, yet the machine does not make an excessive quantity of slime, throwing out the pulp as soon as it is the required size to pass the screen, which always tends to efficient duty. It has been found on the field that, on the average, the slime assays lower than the sand, just the reverse of stamp-mills on the same ore.

The erection of a Huntington mill is very simple. A good concrete foundation is necessary, upon which the 12 by 12-in. wood frame, supplied with all mills, is bolted. The mill bottom is then bolted to the frame, and then the false bottom is fitted in and cemented up. The die-ring is lowered in place and wedged up evenly all around, wood wedges being procured locally from one of the timber mills. The pinion-shaft, vertical spindle, and gear are next fitted and lined up. The housing is bolted on to the bottom of the mill, not forgetting the jointing. The disc driver, which carriers the rollers and scrapers, is keyed to the vertical spindle. The roller shells are wedged on, and the spindles carrying these are fitted into the yokes, which hang from the disc driver. The rollers should be 1 in. from the bottom of the mill, and the top of these should be level with the die-ring. The scrapers are set about ¼ in. from the pan bottom. Mills are made with three or five openings for discharge of pulp, and are fitted with punched screens.

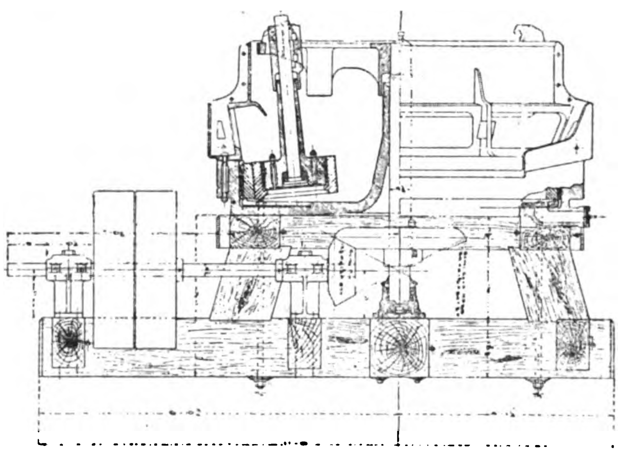
DETAILS OF HUNTINGTON-MILL CONSTRUCTION

Diam. of mill.	Width of frame.	Length of frame.	Height of frame.	Total height of mill.
3½ ft.	49½ in.	79 in.	27 in.	56¾ in.
5 ft.	66 in.	120 in.	34 in.	68 in.
6 ft.	78 in.	132 in.	37 in.	77½ in

Diam. of mill.	Height from floor to discharge.	R.p.m.	Hp.	Weight.
3½ ft.	35½ in.	90	4	8,000 lb.
5 ft.	40 in.	70	6	15,000 lb.
6 ft.	47¾ in.	55	8	20,000 lb.

The preceding table shows the dimensions of Huntington mills as supplied by Fraser & Chalmers.

The mill is a good amalgamator, and on this class of ore, the gold being rather fine, amalgamation varies from 25 to 60%. The mercury fed into the mill gets well mixed with the pulp by the action of the rollers and scrapers, causing a general violent circulation. A short lip plate is generally used. Some plants have a single copper plate about 12 by 5 ft., while others have two or three short plates covering that length, with a well between each. The owners argue better results are obtained by the latter system. The former also appear to be satisfactory, so each works on experience gained. The Huntington need not be fed with as much mercury as a battery, as it is being continually brought into contact with the pulp in circulation.



CROSS-SECTION OF HUNTINGTON MILL

In a 5-ft. mill the pinion or pulley shaft revolves at 140 r.p.m., and the vertical spindle 70. The step-bearing of the latter requires plenty of good oil, and the gear should be well greased, thus prolonging the life for years. The bushing of the bearing, and the steel toe and button do not require renewing very often. The rollers should be regularly oiled. The Nelson feeders are driven by an eccentric or belt from the mill pulley shaft. The average size of feed is 2 in. The ore as fed in is immediately caught by the scrapers which throw it in front of the rollers, which in turn crush it against the die-ring. The four scrapers are placed staggered in the disc driver, and it is important that these be kept in good order, renewing the shoe as soon as worn to any extent, especially the one nearest the die-ring. If the scrapers are not attended to they will not throw up the ore to be crushed. By the sound of the mill the attendant can judge how it is crushing. When fed just right, a rumble is heard; but when overfed, there is a swishing sound, and ore is to be seen scattered over the bottom of the

mill toward the centre. This is bad work, as the rollers drag the ore about,, throw out mercury, and the capacity of the mill falls off greatly.

The die-ring should be of uniform steel, else there will be soft patches in it which cause uneven wear, so much so that they have to be replaced at times. Some of the millmen prefer a small 'hump' to form, arguing that the rollers thus swing about more, giving more impact action, but this scarcely seems correct. The makers supply a grinding tool for turning up die-rings. This consists of a spindle, fitted to a yoke similar to the roller, to which is fitted a segment of a circle the same radius as the die-ring, and about 24 in. long, 6 in. wide, and 1½ in. thick. To this is bolted a shoe, which may be renewed when worn. The grinder is put in the place of one of the rollers, and presses hard upon the die-ring, wearing off the uneven spots, this being done when the mill is crushing. Generally, the apparatus is not used here. It must be a great brake on the mill, and add considerably to the power used, although only in for an hour or so. The millmen apparently consider that when a die-ring wears badly, truing up is only a temporary job, and the uneven wear will occur again, so it is best to renew the ring.

OPERATION OF HUNTINGTON MILLS, KALGOORLIE

Name.	No. of mills.	Size, ft.	Speed, r.p.m.	Hp.
Golden Dream	1	5	68 to 71	10
Cassidy Hill	1	5	70	6
Lone Hand	1	5	68	6
Hannan's Consols	2	5	74	6
Adeline	3	5	70	6
Lomassney & Co.	1	5	74	6
Lake View South	1	5	70	8 to 12

Name.	Drive.	Hp. of engines.	Daily capacity, each mill, tons.
Golden Dream	Crossley gas	32	54
Cassidy Hill	A. E. G. motor	12	50
Lone Hand	A. E. G. motor	12	80
Hannan's Consols	A. E. G. motor	18	30 to 45
Adeline	Hornsby gas	29	30
Lomassney & Co.	Crossley gas	24	48
Lake View South	Tangye gas	25	50 to 70

Punched screens of 250 holes, equal to about a 30-mesh wire screen are used in all cases, being the best to stand the heavy splash of the mill. To prevent sand being splashed up into the hub of the roller head, and getting down into the friction rings, all mills here have a piece of 5-in. iron pipe shrunk on, thus preventing the pulp from splashing up. From the tables given it will be seen that the 5-ft. mill crushes from 30 to 80 tons daily, and tests by the Associated Northern showed 120 tons. The former tonnage is on quartz and the second on ordinary oxidized ore, while the last is on

a soft kaolin with small ironstone seams in it. The table below gives details of the principal Huntington mill plants.

At most of these plants no proper record is kept of costs, so the table is lacking in details. Some definite costs are promised in a few months from the new plant at the North Boulder, and Gimlet South Extended.

The latest Huntington mill plant to be erected in Kalgoorlie is on the North Boulder property. This is a well designed plant, and has been well built. It consists of a grizzly, 7 by 9-in. Dodge breaker, 100-ton bin, Challenge belt-driven feeder, one 5-ft. mill, one long copper plate, and several pumps, all driven by a 35-hp. Campbell gas engine. In the cyanide department are several collecting-vats, and four percolating-vats for the clean sand, the slime flowing to a dam. Costs will be very low here.

Name.	Class of ore.	Water used per ton, gal.....	Life of die-ring days ....
Golden Dream	average oxide	250	240
Cassidy Hill	fair per cent quartz	200	180
Lone Hand	kaolin	250	210
Hannan's Consols	fairly hard	180	160
Adeline	mixed	200	150
Lomassney & Co	iron and quartz	...	...
Lake View South	iron and quartz	200	150-270

The cost of water at these mills is 48 to 87c. per 1000 gallons.

Plates.	Amalgamation.	Life of roller, days.
12 by 5-ft.	30% inside	120
12 by 5-ft.	varies	70
(2) 5-ft.	varies	120
(2) 5-ft.	varies	60
(2) 5 ft.	25 to 60%	120
(2) 5-ft.	20 to 50%	...
12 by 5-ft.	low %	180-270

Costs of milling are given, in cents per long ton, as follows:

Name.	Water.	Power.	Labor.	General.	Total.
Golden Dream	23	5	30	8	66
Cassidy Hill	..	..	..	..	80
Lone Hand	18	8	12	2	40
Hannan's Consols	8	..	..	..	72
Adeline	18	3	..	..	..
Lomassney & Co.	8	8	..	..	..
Lake View South	..	8	..	..	..

Probably the most interesting and up-to-date plant using Huntington mills, and one which must show record costs for complete treatment, is being erected on the Victorious mine at

Ora Banda, 40 miles from Kalgoorlie, by the Associated Northern Co. A description may be of value, and a progress photo of the plant is shown. I had supposed this to be one of largest installations of Huntington mills for primary crushing; but there is a plant of six mills at work at the Zuiho mine in Formosa, crushing 100 tons daily of an ore consisting of slate and limetsone. The Associated Northern has developed 160,000 tons of oxidized ore assaying $5.25 per ton, and was bought for $110,000. To check the sampling a Huntington mill was erected, and several lots of ore crushed, confirmed this. The mill dealt with 5 tons per hour, or 120 tons daily. The ore is kaolin with intermixed seams of ironstone which carry the gold. The 350-ton plant, now in full operation, consists of: One jaw-crusher driven by a Tangye gas-engine; 14-in. belt conveyor to mill storage bins, Nelson feeders; four 5-ft. Huntington mills with 30-mesh wire screens; inside amalgamation but no plates; pulp flows down a cement launder to a large two-plunger pump which elevates to spitzlutte; underflow to two 5-ft. grinding pans, pan overflow and that from the spitzlutte to the thickening-tanks; slime to four ordinary style agitators, then to three new type Ridgway machines for filtration of gold-bearing solution, these to deal with 120 tons each daily; residue to be pumped to dam. Two other Tangye gas-engines drive the main shafting. The water used is salt, and costs 87c. per 1000 gallons. It is expected that the complete treatment of this ore will cost $1.50 per ton. Details of the mill's performance will be given later.

There are a number of stamp-mills at other mines crushing oxidized and customs ore. Of these, the mills on the Hannan's Reward and Hidden Secret are of some note, in that they are fairly well designed. With reference to one of the others, it may be said that five 1000-lb. stamps only crushed 28 tons daily of ore from the Lake View South, as against 50 to 70 tons by the Huntington mill. A few details of these two mills are given below.

As these mills are on customs work at times, costs could not be procured. The Hannan's Reward gets about 50 tons of sand out of 100 tons of ore crushed. The Hidden Secret had a No. 4 Krupp ball-mill working for a time, with a daily capacity of 80 tons through a 25-mesh screen, which compares well with the 100 tons daily crushed by the No. 5 mill at the Chaffers. The wear in the Hidden Secret mill was not very heavy, but for some reason a good extraction could not be secured from the sand. Nine years ago the Eclipse mine, now belonging to the Oroya Links, had twenty 900-lb. stamps crushing the usual class of oxidized ore. These averaged 9 tons daily each, but made a good deal of slime.

In the treatment of the oxidized ore the Associated Northern mill at Kalgoorlie first weighs the ore and takes moisture samples. The ore then passes a No. 5 Gates crusher into bins by distributing belt. Shaking feeders feed the No. 5 Krupp mills, which have a 9-mesh screen, the crushed ore dropping upon a 14-in. belt. Under the mill it is automatically sampled continuously. The belt elevates

the ore to a small mixing pan, where it is mixed with 0.04% KCN solution, and then the pulp flows into three 5-ft. pans, the overflow going directly to agitators. It is agitated for 8 hours, pumped into presses, and washed as usual, the residue being trammed away. The total cost of this complete treatment is $2.25 per ton, being somewhat high on account of the variable tonnages dealt with, common to all custom mills. Manufacturers and text-books on Huntington mill practice cite the following points in their favor, which seem to be pretty well borne out in practice, though perhaps I have failed to prove them.

Reduced first cost, say two-thirds that of stamps; saving in power, or one-third per ton crushed; wear and tear and cost of renewal less; less flouring of mercury; less freight; erection, one-tenth that of stamps; wearing parts easily duplicated; better discharge, and pulp in a better condition for concentrating; better amalgamation; less sliming, as pulp is discharged immediately on reaching the proper size.

DETAILS OF STAMP-MILLING AT KALGOORLIE

Name.	No. of stamps	Weight, lb.	Drop	Per minute	Screen, mesh	Crushed daily, tons	Plates
Hannan's Reward	10	1100	5	114	26	100	1 long
Hidden Secret	5	1200	6	110	25	45	1 long

Name.	Drive.	Treatment of sand.
Hannan's Reward	30-hp. G. E. motor	Not all the time.
Hidden Secret	25-hp. Crossley gas-engine	7 days' percolation with 0.08 and 0.06% solution.

The whole question of Huntington mills may be summarized by saying that if a soft to moderately hard ore is to be crushed, install one; drive it and other machinery with any good suction gas-engine; treat the sand by ordinary percolation methods, and perhaps the slime with vacuum plant. Cheap and good results should follow. This is an admirable arrangement for any small mine which has to deal for a considerable time with oxidized ore only.

CONICAL TUBE-MILL GRINDING

(August 20, 1910)

The Editor:

Sir—The comparatively recent use of the conical mill for fine grinding has brought up many questions, especially among actual users of the apparatus, concerning its ultimate limit of efficiency.

It seems that this type of mill, while possessing certain attractive features of design which should add to its efficiency, has only been tried out grinding a few classes of ore under somewhat similar conditions. While the manufacturers of the different makes of conical mill are confident that it forms a step forward in the art of fine grinding, they are admittedly doubtful as to its limitations. Further and more detailed data are desirable on this subject, and with this in mind I submit the following figures.

It must be understood that the performance of both mills, as shown by the figures, is based on similar operating conditions, both being run on a hard quartz gold ore. The work shown was done at Polaris, Yuma county, Arizona, and the performance of the different grinding devices was carefully watched by the general manager of the company, while certain adjustments were made by him with a view to securing the best possible results from the machines in use.

SIZING TEST ON ABBÈ TUBE-MILL, 4 FT. 6 IN. BY 20 FT., 30 R. P. M.
75 TONS PER DAY

Mesh	Feed. Per cent.	Issue. Per cent.
On 20	3.1	....
" 40	15.5	3.2
" 60	15.6	4.5
" 80	14.6	10.5
" 100	9.6	15.8
" 120	2.4	2.4
Through 120	39.2	63.6

SIZING TEST ON 8-FT. SINGLE CONICAL MILL, 25 R. P. M.
75 TONS PER DAY

Mesh	Feed. Per cent.	Issue. Per cent.
On 20	15.1	3.5
" 40	35.6	21.2
" 60	10.4	14.1
" 80	11.1	17.9
" 100	6.4	11.2
" 120	0.7	1.6
Through 120	20.7	30.5

The conical mill was run at a speed varying from 18 to 30 r.p.m. and experience showed that 24 to 25 r.p.m. gave the best results. An increased speed above this had no beneficial effect, while it probably increased the wear on linings and pebbles and, of course, needed more power. At a speed of 25 r.p.m. about 25 hp. was required. The mill was charged with 3½ tons of pebbles. The Abbé tube-mill was charged with 8 tons of pebbles, which filled it about 6 in. more than half full. Performing the work indicated in the above test 30 hp. was required.

These figures are submitted with the idea that they will prove of interest to millmen in general. It would be helpful to many engineers who are encountering similar problems if other members of the profession would publish accurate data relating to their own experiences.

STUART TODD.

New York, July 29.

SUGGESTIONS ON CYANIDE PRACTICE

By J. B. Stewart

(August 26, 1911)

Several writers have from time to time called attention to the abvisability of separate treatment for natural slime. I think that much may be done in this direction to develop the highest efficiency in many plants. E. M. Hamilton has pointed out that at El Rayo mine a higher net profit per day was obtained by treating very fine sand separate from the slime. The essential reason for this treatment at El Rayo was that whereas the extraction obtained from the sand and slime when collectively treated was just as high as when separately treated, still the time of treatment required for such an extraction reduced the capacity of the plant a great deal. By treating separately, a shorter treatment for the slime gave the maximum extraction, and only a trifle longer treatment was required on the sand. The result was more tons treated per 24 hours with the same equipment and a higher return with no increase of costs. In the same way I believe that the natural slime separated before final regrinding begins, would yield the maximum extraction much quicker. In this direction I expect much advance in reducing tank capacity and volume of solution.

The success reported for the Just process must call for considerable argument in the near future, regarding the relative merits of two theories which seek to explain the best conditions for rapid dissolution of silver values in a slime charge. I would like to see the topic 'Diffusion versus Circulation' discussed. Is it necessary to circulate solution faster or slower than the slime, to promote a perfect diffusion of the dissolved values, from the immediate surface of the slime particles to the general mass of solution? Or can the same rate of extraction, accomplished by the above method, be obtained by merely keeping the slime particles suspended in the solution, depending upon a purely physico-chemical diffusion of the dissolved metals rather than mechanically assisting such an action?

Touching on the subject of the economic limit for moisture in the feed for a tube-mill, it is only necessary to note the difference between the consistence of sand and slime, each containing 25% moisture, to realize that whereas the sand might grind all right at that dilution, certainly the slime would not do so. As the surface of the material increases due to reduction in size of its grains, the amount of moisture required to keep it mobile necessarily increases. This points to the advantage of stage grinding in which, for instance, ¼-mesh product could be introduced at 20% moisture in the first stage and after classifying the discharge, the second stage could be conducted with 39% as suggested by Walter Neal's experience at Dos Estrellas.

Next, as to stage grinding, I wonder that the advantages of this refinement have not made greater headway in general practice,

and I also wonder why the high-speed Chilean mill has come into favor instead of a short-length tube-mill as the intermediate grinding unit, in several large companies which have already adopted stage grinding. Certainly the operating cost for the Chileans must be much greater than it woudl be for 15-ft. tube-mills. In fact, I shall consider the tube-mill more efficient in power, duty, labor, and supplies than the Chileans.

In conjunction with stage grinding as already suggested, it would seem preferable to use the Callow screen to separate the coarse feed for the primary grinders and Dorr classifiers for the secondary mills. Whereas there can be no doubt of the efficiency of the cones provided with diaphragms, as described by Mr. Neal, for dewatering a tube-mill feed, still I doubt whether they lend themselves so well as the Dorr classifiers do, to re-handling the entire discharge of the tube-mill. The Dorr machine is a classifier and a dewatering machine as well, with no rival for compactness and saving in grade or elevation about the plant. On the other hand, the dewatering cones using baffles must be preceeded by other cones for separating the greater bulk of slime, and this entails much more difference in elevation.—*Inf. y Mem. del Ins. Mexicano de Min. y Metallurgia.*

GIESECKE TUBE-MILL

(September 30, 1911)

Several of the leaders of the mining industry on the Rand have gone into ecstasies lately over the adoption of the Giesecke patent tube-mill, evidently under the impression that it was likely to revolutionize milling operations. It is even claimed that it may result in the total elimination of the gravity stamp, as by its use ordinary broken ore is easily reduced to such a fineness as to readily pass a 200-mesh screen. The mill is a German invention and has been hitherto principally used in the manufacture of cement. It may be described as a modified tube-mill some 24 ft. long, one-fourth of which is 7 ft. 6 inches in diameter and the remainder about 6 ft. in diameter. Its capacity is 50 tons of pulp, weight 68 tons, and when in action the mill revolves at the rate of 25 r.p.m., a 400-hp. motor being used to drive it. As far as can be gathered, it appears to be crushing 350 tons per day of ore broken previously into various sizes up to 6-in. cubes, with the result that 81% passes through 200-mesh, 6% over 200, 7% over 120, 5% over 90, and only 0.5% over 60-mesh. The method of working is to feed into the larger end of the mill, the coarse particles being confined to this until sufficiently reduced in size to pass through a screen dividing the large diameter portion from that with a 6-ft. diameter. The reduction is effected by steel balls varying in diameter from 2½ in. to 4 in. diameter in the large or breaking chamber, and down to 1¼ in. diameter in the grinding portion of the mill. As already shown, the pulp leaves the mill in a remarkable condition of fine-

ness, the coarser particles being returned to the mill for further attrition. Water to the amount of less than a third in weight of rock treated is required, and as the mill is fitted with automatic lubricating devices, it seems to require a minimum of personal attention. The capital cost of such a mill is stated to be £6000 net, without including buildings, erection, or motive power. It is estimated to equal in capacity a 50-stamp mill, but so far no idea can be given as to the actual running costs. From what can be gathered, these are not lower than the lowest on the Rand—say, for instance, at the Roodeport United mine, where, with heavy stamps, a half-inch screen is used and 20 tons per stamp per day crushed, the pulp being finished by ordinary tube-mills at a total cost of 2s, 6d. per ton. 2 Considering the reputed duty, the capital cost is low, but the question of most concern is the efficiency and the actual cost per ton treated. These do not seem to have yet been fully ascertained. When the design of the mill is taken into consideration, it is evident that the cost of maintenance will be large. At all events the experiments being made with the mill under local conditions by a committee of the Transvaal Chamber of Mines were for some time suspended because the steel balls used inside the mill became too worn to be effective, and the local agents and patentee did not seem to have any spares on hand to replace the worn ones. It would thus appear that the wear and tear was greater than anticipated, and to have over 20 tons of steel balls continually undergoing wear and tear must in itself constitute an expense far more than the total cost of running an ordinary tube-mill, not to mention the cyanide consumption this iron would cause. Other parts of the mill also do not seem able to stand the severe and prolonged strains thrown upon them. Before it can be said to justify the good opinions formed concerning it, some important vital alterations may become necessary. Even with such improvements it is doubtful whether the stamp-mill will be dispensed with, and as to reducing the ore to the necessary fineness of pulp at one operation, there are grave doubts among milling engineers here as to whether that is possible.

RE-GRINDING SAND

(April 29, 1911)

The Editor:

Sir—I should be pleased if any of the readers of the *Mining and Scientific Press* would furnish data as to the tonnage of sand (that is, ore crushed in a stamp battery through 30-mesh screens, and the slime removed) that can be ground to pass 150 mesh in 24 hours by a Wheeler grinding pan five feet in diameter. Also I would like to know the horse-power required to do the work. There exists here a great diversity of opinion as to the performance of the ordinary grinding pan such as is in common use in Australia and elsewhere.

EX-KALGOORLIEITE.

Mexico, March 24.

(September 30, 1911)

The Editor:

Sir—The inquiry of 'Kalgoorlieite' as to the duty of Wheeler pans, has remained long unanswered. He will find exhaustive data on the subject in R. Allen's 'West Australian Metallurgical Practice,' together with screening analyses with various feeds. In brief, the average duty of a 5-ft. pan is about 20 tons per day as a stage-grinder, or about 8 to 10 tons per day as a slimer. The power consumption averages 6 horse-power.

'Kalgoorlieite' is evidently surprised at the diversity of opinion existing in Mexico with regard to the efficiency of the grinding-pan, a diversity that needs some explanation in view of the fact that, at Kalgoorlie, pans are in general use and are considered thoroughly efficient machines. 'Kalgoorlieite' has my sympathy if he has been endeavoring to defend, in Mexico, the practice of pan-grinding, since he will never convince the majority. The main reason for this is that metallurgists in Mexico have nothing to learn with regard to tube-mill practice. The second reason is that pans have never had a fair chance there. The Wheeler pan, as used at Kalgoorlie, is practically unknown in Mexico. In its place is the standard amalgamating and grinding pan. The latter was in general use for intermittent charge-grinding in amalgamating plants before the cyaniding of silver ores was practiced. In several cases it was adopted as a re-grinder, preparatory to cyaniding; but failure invariably resulted. Personally, I have done everything in an endeavor to coax even a moderate duty out of this type of pan, but all to no purpose. The general design was hopeless, the grinding surface was insufficient, the 'spider' taking up most of the room; the shoes and dies were much too light, and there was no satisfactory method of compensating for the loss of weight due to wear. The power consumption was in excess of that needed for an efficient pan of the same size, but the maximum output I could get on ordinary quartz ore and with the pan fitted with new shoes and dies and a Wheeler discharge was about 3 tons slimed per 24 hours, from 20-mesh.

The conversion of this type of pan into an efficient machine meant a complete renewal of everything, with the exception of the shell and gears, and, although I thought otherwise at first, I soon came to the conclusion that investment in tube-mills, and the scrapping of the pans, was the best course to pursue. To judge from the number of these machines that one sees on junk piles in northern Mexico, I was not alone in my experience or conclusions, and I gather that this is one of the reasons why grinding-pans are not in favor in Mexico.

A. W. ALLEN.

London, September 11.

PEBBLE EFFICIENCY IN TUBE-MILLING

By A. W. Allen

(January 6, 1912)

The question of the efficiency of pebbles of various sizes in tube-milling is an open one. In the earliest application of these machines for sliming gold ores at Kalgoorlie it was a common practice periodically to overhaul the contents of the mills and to reject the smaller flints. Some advantage evidently accrued, although the question was not discussed, and there was a want of unanimity of opinion as to the advantages gained by the operation.

The bulk of the sliming action in tube-mills results from the direct impingement of the pebbles, one upon the other. This is also the easier action to consider theoretically. If it be supposed that the contained pebbles in a tube-mill are perfect spheres of 1, 2, 3, and 4 in. diam., their respective masses are in the ratios of 1, 8, 27, and 64; and since the velocity at point of impact would practically be the same in each case, the available energy can be roughly estimated as follows:

(1) At point of impact,

Diam. of pebbles.	Momentum ratio.	Energy ratio.
1 inch	$1v$.	1
2 "	$8v$.	8
3 "	$27v$.	27
4 "	$64v$.	64

(2) Assuming one point of impingement per pebble, in each cubic foot,

Diam. of pebbles.	Points of impingement.
1 inch	1728
2 "	216
3 "	64
4 "	27

(3) Energy ratio times frequency equals composite efforts per unit volume,

Diam. of pebbles.	Energy ratio.	Frequency.	Unit of effort.
1 inch	1	1728	1728
2 "	8	216	1728
3 "	27	64	1728
4 "	64	27	1728

In place of a statement of simple fact the above details may seem labored, but the idea is to emphasize the utility of the smaller, as well as the larger pebbles. A composite equality of available energy per unit volume does not, however, necessarily imply identical performance with pebbles of any size, under all conditions. Fracture, and resultant effect, will depend on the

friability of the ore as compared with the size of the particles to be broken and the momentum of the mass that is expected to break them. Present-day tube-mill practice indicates that a cylindrical mill operates with a mixture of pebbles of all sizes below 4 or 5 in. diam., with the exception of a proportion of the smallest, which are periodically discharged with the pulp. Attention is seldom paid to the question of feed, but it is evident that efficiency is sacrificed if the material fed contains a proportion that is too coarse even to be fractured by the smaller pebbles; that is, energy is being lost if any individual impact is unproductive of result.

Under these conditions a reduced feed often indicates an improved efficiency, the probable reason being that, with the full feed, coarse sand is found even at the discharge end of the mill, and, in consequence, the smaller flints have no opportunity for doing useful work on particles of ore proportionate to their size. By lowering the feed the coarse sand may be uniformly reduced in the first half of the mill. The smaller pebbles in the second half then have an opportunity of doing useful work in return for power expenditure. From which may be inferred that an efficiency with regard to feed-volume is strictly relative to the grade of ore being introduced.

In considering the action of the larger pebble on a finer grade of material the area of impingement is proportionately increased after fracture of the largest sized particles occurs; and any unspent energy remaining is concerned with the pulverizing of a larger area of smaller particles. The energy expended by the larger pebbles can never be unproductive as long as the unclassified product from the mill contains a proportion of unslimed ore.

Tube-mills are used for sliming a product previously milled, and also for re-grinding a coarse battery product preparatory to classification and leaching treatment of the remaining sand. Where pebbles, and not rough cubes of the ore itself, are used for grinding purposes, it would seem that the tube-mill is better adapted for sliming than for re-grinding; and that where the output from the batteries is through a coarse mesh, another class of pulverizer or grinder should intervene between them and the tube-mills in order to handle and reduce the proportion of coarse sand produced.

TUBE-MILL PRACTICE AND LINERS

By F. C. Brown

(February 3, 1912)

It is well known that the tube-mill came into the mining industry from the cement industry, and in adapting the machine for the wet-grinding of ores very little change was made in it, considering the very different class of work it was called upon to perform. In cement work the mills are usually of large diameter and the thick

silex lining used does not take up an unreasonably large percentage of the working area. This type of lining has a very long life in dry grinding, and the danger of silex blocks working loose is not great.

In wet grinding the conditions under which tube-mills are operated very tremendously, and the mills and their method of operation require modification accordingly. In some cases the feed is very coarse, and medium or very fine grinding is required, while in other cases the feed is already very fine and the tube-mill is required to produce slime. Some mills are operated with a large quantity of material passing through them, the coarse, or not sufficiently ground particles, being continuously returned to the feed for further grinding; while other mills have a small feed and the material only passes through the mill. One would naturally infer that, in order to meet such a variety of conditions, the following factors in tube-mill practice would have to be carefully considered:

a. Diameter of the mill.
b. Length of the mill.
c. Speed of the mill.
d. Size of the pebbles.
e. Load of pebbles.
f. Percentage of moisture in material being ground.
g. Lining of the mill.

a. Diameter of the Mill.—I have made careful tests with regard to this, and have come to the conclusion that it is a waste of power to use mills of large diameter when the material to be ground is already fine; my idea being that a mill of small diameter and small pebbles are the most economical for such work. For grinding the pulp coming from stamps crushing through wire screens having, say, six holes per linear inch, I have found mills of 4 ft. diameter inside the liners suitable. When finer crushing is done by the stamps, smaller diameter mills would answer as far as economy in grinding is concerned, but such small mills are inconvenient when it comes to relining and repairs. If it is a question of grinding a very coarse feed, say particles up to ⅓-in. diameter, I prefer a fairly large diameter mill, but even for this work a diameter of 5 ft. inside the liners is large enough, provided there is a suitable arrangement for returning coarse particles to the mill for further grinding.

b. Length of the Mill.—For grinding 6-mesh pulp from stamps so that it will pass 200 mesh, mills of 4 ft. diameter and 16 ft. long answer well. At the Broken Hill Proprietary mine of New South Wales, mills 4 by 13 ft. are doing good work in regrinding tailing that will pass 10 mesh and leave 60% on 40 mesh; the finished product being of a fineness of about 10% on 40 mesh. One of the main objects in this case is to produce a minimum quantity of slime, and it is quite likely that mills of less than 13 ft. in length would do even better work.

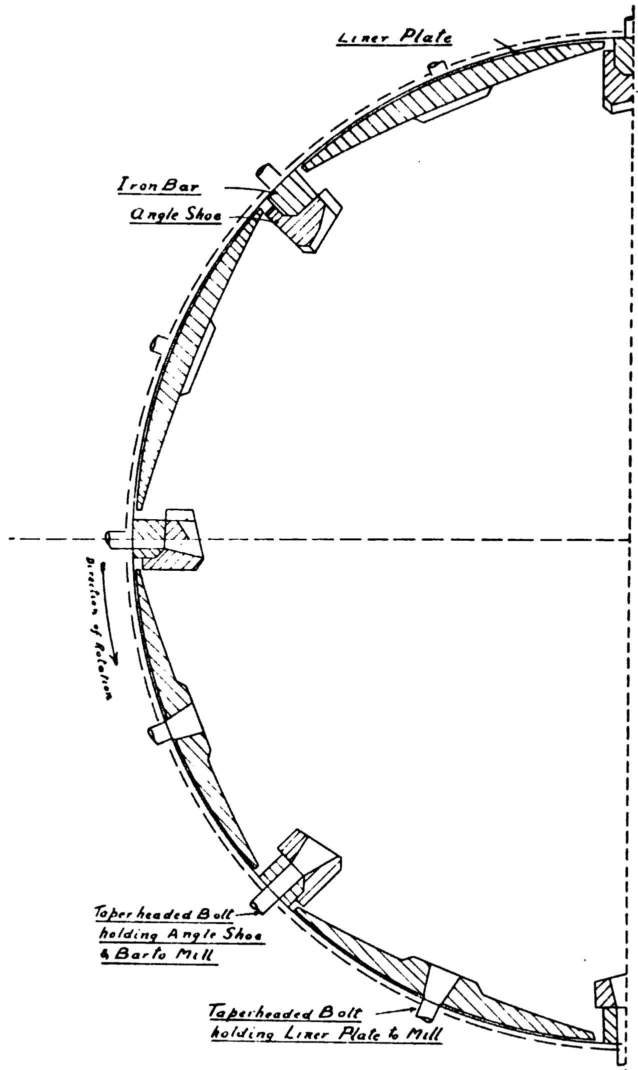
Speaking generally, the length of the mill should be determined by the nature of the finished product required. If the object is to slime everything, the mills should be long, but if a granular or sandy product is required, they should be short; and in all cases they should be operated in conjunction with some efficient system of returning insufficiently ground particles to the feed. Two mills run in tandem with a classifying device between them will do better work than one mill of the same diameter with a length equal to the combined lengths of the two mills.

c. **Speed of the Mill.**—This is one of the most important factors in operating tube-mills, and depends upon the diameter of the mill and the type of lining used. With thick lining like silex blocks it cannot possibly be correct during the whole life of the lining, as, in a 4-ft. diam. mill, these blocks when new take up 16 to 20% of the cubic content of the mill, according to their thickness. The usual speed for a mill 4 ft. in diam., lined with a smooth liner such as silex blocks or metal plates, is 32 r.p.m., and for a 5-ft. mill, 27 r.p.m. If these mills are lined with metal plates it means taking off 2 in. from the diameter to arrive at the working diameter, which would be 3 ft. 10 in. and 4 ft. 10 in., respectively, giving peripheral speeds of 382 and 407 ft. per minute. With silex liners these speeds would be considerably less, depending on how much the liners are worn. In an article by H. Standish Ball, 'Economics of Tube-Milling' (*Mining and Scientific Press,* September 23, 1911), the most efficient peripheral speed for a mill 2 ft. 10 in. diam. inside the liners is given at 333 ft. per min. (37 r.p.m.), and in the same article he gives the speed of the standard tube-mill used in the Transvaal, which is 5 ft. 6 in. diam. inside the shell, at 31 to 32 r.p.m. Such a mill lined with silex blocks would have a working diameter of 4 ft. 6 in. when the liners are new, and 5 ft. when they are worn and ready to be replaced, giving peripheral speeds of 445 ft. and 494 ft., respectively. In my opinion, the mill is running at too high a speed and the variation of speed due to the wear of the liners shows the unsuitability of silex blocks as liners.

d. **Size of the Pebbles.**—This should bear some ratio to the diameter of the mill and the kind of grinding to be performed. A mill that has been in operation for some time contains a good percentage of small pebbles, so the pebbles being fed to the mill can be larger than would be correct for a full charge. Pebbles ranging from 2 to 3 in. diam. answer for the usual grinding of pulp coming from stamps. For grinding coarse material, pebbles up to 5 in. diam. are suitable.

e. **Load of Pebbles.**—There is a great variety of opinion regarding this, but the usual practice is to keep it about up to the axis of rotation when smooth liners are used, although it appears that in the Transvaal the pebble-load for a mill 5 ft. 6 in. diam. is kept at 3 in. above the axis of rotation, according to Ball.

f. **Percentage of Moisture.**—This is a factor that is influenced by so many conditions that each mine has to find out by experiment what answers best. The percentage of moisture should



KOMATA LINER

range from about 38 to 50%, and depends on the specific gravity, coarseness, and nature of the material to be ground. The percentage of moisture has a very marked influence on the grinding, and the eliminating of slime from the feed is also of great importance.

g. Lining of the Mill.—The first mills used for ore were lined with either silex blocks or smooth metal plates, both of which fall far short of being an ideal lining, as the former takes up too much space in the mill, and the latter allows the pebbles too much back slip. The improved liners are the Honeycomb, the El Oro, and the various forms of corrugated metal liners, all of which have their distinctive feature of merit, and I have recently introduced in this country a ribbed liner known as the 'Komata' liner (named from the Komata Reefs mine of New Zealand, where it was first used). This liner is now used by most of the miners in New Zealand and has been adopted by the Goldfield Consolidated Mines Co. of Nevada, the Tonopah Extension Mining Co. of Tonopah, and the Associated Milling Co. of Manhattan, Nevada. It has the very important feature of only slightly altering as regards thickness and shape during its whole life, is easily and quickly put in place, has a remarkably long life, and shows a greater grinding efficiency and decreased consumption of pebbles compared with other liners. The nature of this liner is shown by the accompanying sketch. Its adoption requires an alternation in the speed of the mill and also in the load of pebbles. The former should be about 330 ft. per min., peripheral speed, for mills of 4 to 5 ft. diam., and the latter should be 5 to 7 in. below the axis of rotation, depending on the diameter of the mill. These figures are deduced from New Zealand experience, and it will be interesting to see what the results are from tests made in this country. The Edgar Allen American Manganese Steel Co. has the manufacturing and selling rights for the Komata liner.

There are several theories regarding the exact action of the pebbles in a tube-mill which produces the grinding of the sand or ore, some claiming direct impact or impingement as being the most important factor, while others lay stress upon the rubbing or abrasive action. In my opinion, it is difficult to come to any exact conclusion, but viewing the matter from a strictly practical standpoint. I lay the greatest stress upon the *movement* of the pebbles, my experience being the more movement the more grinding; hence the advantage of the Komata liner, as this permits of no 'dead' place in the mass of pebbles and ore.

Direct impact is certainly not the only factor in grinding, as I have fed $^1/_8$-in. diam. particles of hard quartz to a 4-ft. diam. mill and they were discharged slightly rounded after the first passage through the mill, but being several times returned, they gradually become converted into slime, and no doubt very fine slime, just what is required with some ores for high extraction. This might be an explanation of the fact that the addition of tube-mills to a plant, even if the ultimate fineness of the grinding, ascertained by sieve tests, is about the same as before the tube-mills were used, often shows a remarkable increase in extraction. It has always seemed good practice, from my experience, to have a certain amount of very coarse sand continuously going to a tube-mill along with the feed, as this materially increases the grinding efficiency.

THE ECONOMICS OF TUBE-MILLING

By H. Standish Ball

(September 23, 1911)

*The modern tube-mill is a most efficient grinding machine, and is rapidly taking the place of Chilean mills, roller mills, grinding pans, and similar apparatus. Though used for many years as a cement grinder, the first reference found to its use as an ore-crushing machine was in 1899, when it was used by L. H. Diehl for grinding Kalgoorlie ores. For the next few years very little was heard about it, nor was it till about 1905 that it became a machine of great industrial importance on the Witwatersrand. During the last six years the development of its working details has been rapid, for it was but a step from the iron ball of the ball-mill to the flint pebble, another from the one manhole in the shell centre to two at opposite sides, and one more from the iron lining to the silex, set flat in cement, then back to iron or steel plates, and from these to the honeycombed and ribbed lining. Where formerly silex linings lasted three months and required three weeks for renewals, the life is now twelve months, and replacements are made in two eight-hour shifts. At first the mills were laboriously charged by removing a single manhole cover and throwing in loads of pebbles; next a bin was built over the mill to chute the pebbles rapidly into the mill and thus save time. Such a means as a fixed spiral in the discharge end, or a feeder large enough to pass the pebbles into the feed end of the mill, has materially lessened the old troubles, and tube-mills can be run 97% of the possible running time. With the simplified method of feeding now adopted, the modern operator is gradually dispensing altogether with the flint pebbles, their place being taken by the lumps of ore itself in Rand practice; thus the cost of the pebbles is saved and incidentally the capacity of the plant is increased 1 per cent.

Before giving an example of the process followed out, a few details of the present day tube-mill will be interesting, the mill taken being one of a series used by the Eckstein group on the Witwatersrand. The present standard size mill is 22 ft. long by 5 ft. 6 in. diam. Silex liners are used, 6 to 7 in. thick when new; thus the internal diameter of the mill when lined is about 4 ft. 6 in. The internal volume of a newly-lined tube-mill is 350 cu. ft., the volume of the shell without the lining being 523 cu. ft. The pebble load is a varying factor increasing throughout the life of the liner; with new liners the pebble load is 8 to 9 tons, with worn liners (liner 3 in.) the load is 12.8 tons, and may reach 14 tons just before the liners are removed. When the mill is stationary, the level of the pebbles is 3 in. above the axis of the rotation of the mill; with new liners the volume occupied by the pebbles is 200 cu. ft. Ore pebbles are now used to a large extent, and the ore is roughly sized to 4-in. cubes before being fed in.

*Abstracted from the *Bulletin* of the Institution of Mining and Metallurgy.

The capacity of this mill is still a debated point, but the following result of one month's working on a typical Rand installation is interesting, the battery screens used being 12 mesh per linear inch. The equivalent of 400 tons of dry sand is passed through the mill in 24 hours, the pulp containing 38.5% moisture. In order to get the best results from the plant in question, the mill should produce 130 tons of product to pass through a screen of 90 holes per linear inch per 24 hours.[1] In addition to the tonnage of sand fed in, there is a weight of 7 tons of banket ore used per 24 hours in place of pebbles. One 125-hp. electric motor is required per tube-mill, though only about 75 to 90 hp. is actually absorbed when once the mill is running uniformly. The number of revolutions per minute is 31 to 32.

As an example of the rapid installation of tube-mills in the Transvaal during the past year, it is interesting to note that whereas, in 1909, 136 tube-mills and 9580 stamps were installed, in 1910 the former were increased by 39.7%, against an increase of 1.6% of the latter. Modern tube-mill plants in other parts of the world differ little in their principle of working, only having variations in minor details; for example, in American and Australian practice a product to pass through a screen of 200 holes per linear inch is frequently required.[2]

The function of the tube-mill as a fine crusher in connection with stamp-milling and cyanidation can be best shown by giving a typical crushing circuit, the one chosen being that adopted by some of the mines of the Gold Fields companies on the Rand. With this system only those particles of the tailing pulp which are sufficiently light to be carried over the hydraulic classifier pass away direct to the cyanide plant, the remainder being diverted to the tube-mill. From the stamps the pulp is taken by means of a pulp elevator to a cone classifier which separates the slime from the sand, the former going direct to the cyanide plant, while the latter is conveyed into the tube-mill by means of a spiral conveyor. From the mill it passes over amalgamated plates to a second cone classifier, where it is re-classified. The number of primary classifiers per tube-mill has been reduced to one, without interfering with efficient classification. The cones used are 6 ft. diam. by 9 ft. deep and contain two diaphragms 14 and 3½ in. diam., the upper one being 5 ft. 5 in. from the top of the cone, and the lower 1 ft. 3 in. from the bottom; the purpose of these diaphragms is to provide a means of obtaining a uniform feed of the pulp. The diameter of the outlet to the classifier is 2 $^{3}/_{16}$ in. and its capacity is about 400 tons of sand with 26% moisture per 24 hours.

At a comparatively early stage in the development of the tube-mill it became evident that high efficiency could only be obtained by feeding a sand containing a minimum of slime, and by reducing the percentage of water below the 50% first taken as a standard.

[1] Screen equal, approximately, to 80 mesh I.M.M. screen.
[2] Screen equal, approximately, to 200 mesh I.M.M. screen.

This demand has resulted in the invention of a number of classifiers and thick-pulp feeds, of which the Caldecott diaphragm cone, mentioned above, is the most satisfactory. Up to the present, however no efficient means has been found for providing an exactly uniform mixture of coarse and fine sand in the mill feed. The diversion of the fine ore from the bins directly to the tube-mill, without passing it through the battery, is one of the best suggestions yet put forward for increasing the efficiency of the stamp-tube-mill combination, and a start in this direction has already been made, with satisfactory results. It is remarkable to what a great extent fine battery crushing has given way to coarse, and it is possible that stamps may ultimately be superseded by other types of coarse crushers when the ore is to be finally crushed in tube-mills.

There are two kinds of crushing which take place inside a mill: (*a*) Crushing by shock, due to the impact of pebbles. (*b*) Crushing by abrasion, due to the rolling and rubbing of the pebbles, and contact with the lining of the mill. One form of crushing or the other will predominate, according to the type of mill used, the pebble load, and other conditions. Although the consideration of the tube-mill from a mechanical point of view is fairly simple, much experience is necessary in order to get the best results from it. For this reason attempts have been made to determine the most economical conditions under which tube-mills can be used, taking into consideration the desired fine grinding as the end to be obtained with the least consumption of power and smallest wear on pebbles and lining. These three points seem to depend on the following four factors:

1. The pebble load.
2. The tonnage dealt with, and the character of the sand.
3. The thickness of pulp.
4. The diameter of the mill and the number of revolutions.

A change in any one of the above will cause a change in two, and sometimes all three of the points given.

The investigation of these interesting points is obviously desirable, and the subject was an eminently suitable one to be undertaken by a laboratory such as that possessed by McGill University, owing to the freedom from interference with continuous tests which would be inevitable on ordinary commercial plants. The rock chosen to be experimented on was an elaeolite or nepheline syenite, of a hard uniform character, composed essentially of orthoclase, elaeolite, and hornblende, and was obtained from the Outremont quarries, Montreal. This rock was used, as it was considered to be typical of a hard, compact ore.

Four investigations were decided on, as follows:

Series I.—In which the feed was to be varied, the moisture, pebble load, and revolutions per minute remaining constant.

Series II.—In which the moisture was to be varied.

Series III.—In which the pebble load was to be varied.
Series IV.—In which the pebble speed was to be varied.

The other three factors remaining constant in each case. For the details of these tests, which were made in a 3 ft. 5 in. by 4 ft. 8 in. tube-mill constructed from an old chlorination barrel, reference should be made to the original paper, as given in No. 83 of the

GENERAL SUMMARY OF TESTS.

Remarks.	Test.	Pebble load.	Pebble vol.	Rev. per min.	Peripheral speed min.	Moisture. %.	Feed. Tons per 24 hrs.	Hp.	Work done per unit.	Net output "−120 grade" (tons 24 hrs.)	Rel. mech. eff. per hp.
	P	lb. 1200	0·5	41	369 ft.	38·0	7·2	6·8	3·89	2·27	4·45
Feed tests	C	1200	0·5	41	369 „	38·0	9·6	5·0	3·36	2·34	6·45
	B	1200	0·5	41	369 „	38·0	12·6	5·1	2·82	2·95	7·02
	O	1200	0·5	41	369 „	38·0	14·4	5·6	2·91	3·31	7·48
	*A	1200	0·5	41	369 „	38·0	18·6	5·8	2·58	3·96	8·29
	R	1200	0·5	41	369 „	38·0	23·0	6·6	2·09	3·79	7·26
Moisture tests	D	1200	0·5	41	369 ft.	30·0	12·6	5·7	2·99	2·93	6·61
	*B	1200	0·5	41	369 „	37·7	12·6	5·0	2·82	2·95	7·11
	E	1200	0·5	41	369 „	50·0	12·6	6·2	3·06	2·95	6·16
	F	1200	0·5	41	369 „	58·0	12·6	6·0	2·84	2·97	5·96
Pebble load tests	*G	900	0·37	41	369 ft.	38·0	12·6	4·9	2·75	2·65	7·07
	B	1200	0·50	41	369 „	38·0	12·6	5·1	2·82	2·78	7·02
	H	1500	0·63	41	369 „	38·0	12·6	6·9	3·31	3·40	6·04
Speed tests	T	1200	0·5	33	297 ft.	38·0	7·2	6·8	4·19	2·45	4·48
	*Q	1200	0·5	37	333 „	38·0	7·2	5·1	4·17	2·59	5·89
	P	1200	0·5	41	369 „	38·0	7·2	6·8	3·89	2·27	4·45
	S	1200	0·5	46	414 „	38·0	7·2	7·5	4·08	2·48	3·87

* Most efficient test of series.

Bulletin of the Institution of Mining and Metallurgy, August 30. The following summary will be of interest:

Feed Tests.—From the results obtained it was found that a feed of 18 tons per 24 hours is the most efficient one for the laboratory mill, any larger feed apparently causing the crushing efficiency, as well as the relative mechanical efficiency, to decrease. This is a most interesting point, as a critical feed is indicated, above which it seems probable that the fine-grinding properties of the mill decrease in proportion approximately to the increase of feed. With a feed of 23 tons per 24 hours, only 3.79 tons of — 120 grade per 24 hours were obtained, whereas the same fine-grade output would be obtained by crushing only 18 tons through a similar

period; this fact shows that the critical feed has been passed between these two amounts.

The theory of a critical feed may have an important bearing on modern practice, as it would seem that more efficient working, with a greater output of slime, would be obtained if a smaller feed were used. In South Africa the tendency at present is to increase the feed in the 5½ by 22 ft. mills, and although they are now feeding at the rate of 400 tons per 24 hours, the belief is held that even heavier feed would result in increased grinding. The above results show the danger of carrying this too far. Using these figures as a basis and roughly calculating the feed for the laboratory mill, it appears that a feed of about 25 tons per 24 hours should result in more grinding than with a smaller feed, whereas the experimental results obtained in these tests indicate that this feed is less efficient than a smaller one and that the most efficient feed for this mill approximates 18 tons in 24 hours. Admitting the discrepancy of comparing two mills differing so radically as to size, it is at least of interest to note that, if any analogy can be drawn, the indications are that the South African mills are overfed and that a feed of about 300 tons instead of 400 tons per 24 hours might result in both increased grinding and efficiency.

Moisture Tests.—The most prominent feature of these tests is the great efficiency of 37.7% moisture over all others. This fact coincides remarkably well with tests carried on in other parts of the world. In those conducted by Walter Neal,[3] results seem to prove the fact that the ideal dilution is in the neighborhood of 39% moisture, although he inclines to the belief that this critical point will vary with the nature and size of the mill. The results of these tests tend to disprove this belief.

Sherrod, experimenting with tube-mills at the Guerrero mill, Real del Monte, found that the grinding efficiency increased with the percentage of solids in feed, up to about 55 or 60%, corresponding to 45 to 40% moisture. G. O. Smart[4] claims that in the tests carried out on the Rand he found that 38.5% moisture is the most efficient, while H. W. Fox[5] discovered that, taking the power consumed in conjunction with the fine-grinding, the most efficient moisture would be 39.6%. It will be noticed that all these critical points are from 1 to 2% higher than the one found by the author, but this seems to be explained by the fact that the sands used, being 18 mesh, were of a finer nature than those commonly used in practice. It has been suggested by J. W. Bell[6] that the critical moisture point depends upon the percentage of voids in sand, and that in all probability when the percentage of voids in the sand inside the mill, the moisture is critical. This theory is of interest in that it indicates that sand-feeds with higher or lower percentages of voids may have higher or lower critical moistures.

[3] *Min. and Sci. Press,* April 2, 1910.
[4] *Jour. Chem., Met. & Min. Soc. of S. A.,* May 10, 1910.
[5] *Mines and Minerals,* June 1908.
[6] *The Mining Magazine,* April, 1911.

The second interesting feature of these tests was the use of the 'transition samples' for obtaining intermediate moisture efficiency. This appears to be a practicable method of obtaining a number of points of efficiency and power at various intermediate quantities of moisture, by simply running a long continuous test and at equal periods changing the moisture without stopping the mill. In the compilation of data and curves this should be of great service both as a check on the work and for effecting a substantial economy in time and labor.

Pebble Load Tests.—The result from these tests indicated that the most efficient pebble load is about $^7/_{16}$ of the volume of mill (or for this mill 1030 lb.), whereas, disregarding power, the most effectual 'crushing load' is about 0.6 of the volume; which corresponds to present Rand practice. Whether it would pay to sacrifice power and gain fine grinding, or vice versa, would, of course, depend upon conditions at the plant.

Speed Tests.—These tests were most interesting, as they proved conclusively that 37 r.p.m., at a peripheral speed of 333 ft. per minute, was the most efficient, both in relative mechanical efficiency and grinding efficiency, thus demonstrating that at that speed the grinding was decidedly better, and the power required less. It was seen that if this speed of 37 r.p.m. is exceeded or decreased, the crushing effect of the pebbles is much reduced, while the power ncessary to drive the mill is increased. In connection with this efficient speed, it is interesting to compare it with that obtained from various formulæ.

Davidson gives as the best practical speed $N = \frac{300}{\sqrt{D}}$ in inches, while White derives a formula $N = \frac{34.22}{\sqrt{D}}$ in metres. (N = speed, D = diam. of mill.) Richards gives 38 r.p.m. as the best speed.

The results are set forth below:

Davidson.	White.	Richards.	By experiment.
34	39	38	37

Thus the result obtained from actual experiment corresponds very closely to the mean of these results.

Calculation of Data.—It is my personal belief that there is a possibility of deducing from the curves obtained from the different tests the probable results that would be derived by running the mill under different conditions. An example of this was shown in the feed tests, where the probable data for Test A were deduced from the curves of Test B. The conclusion from this theory appears startling, for it seems that if one complete set of tests is run with a constant feed, the other factors being varied and the respective curves obtained, it is possible to deduce from them all the necessary information regarding any feed, provided that one test at that particular feed had been previously carried out. As a case in point, it was desired to know what the relative mechanical efficiency for a feed of 7.2 tons per 24 hours would be, with a

moisture of 35%, speed of 37 r.p.m., and pebble volume 0.5, a test with 38% moisture having already been run on this feed. The moisture efficiency curve for Test B was used, and the following data extracted:

Moisture, %	Feed, Tons.	Efficiency, %	Difference, %
38	12.6	7.02	
38	7.2	4.45	0.27
35	12.6	6.75	
35	7.2	x	

$$\text{Hence } \frac{027}{1} \times \frac{4.45}{7.02} = 0.18$$

$$\therefore = 4.45 - 0.18 = 4.27.$$

By actual experiment the efficiency was found to be 4.32, a difference of 1.1% between the theoretical and the actual result.

Transition Samples.—The calculation of intermediate points of efficiency by the method of transition samples and power readings is of great interest, as the efficiency and power required can be found for any intermediate point of moisture, feed, etc., between the limits of any two tests. This period was found with great accuracy by means of nine tests, the time required being checked by both power readings and screen analyses.

SUMMARY

The more important features disclosed in this investigation may be summed up as follows:

(1) The determination of a rate of feed, which if increased or decreased caused the efficiency of the mill to diminish in a marked manner, and consequently may be defined as a critical feed rate.

(2) The determination of a well-defined critical percentage of moisture in the feed.

(3) The determination of a critical speed for the laboratory mill (34 in. diameter).

(4) The determination of a critical pebble load for this mill.

(5) The determination of the length of time required by the mill to assume a uniform condition following a change in adjustment. This is defined as the transition period of the mill.

(6) The substantial corroboration of the author's hypothesis that samples and power observations taken at intervals during the transition period may be used to determine the efficiency of the mill for the calculated conditions at the times the samples were taken. These samples are called transition samples in contradistinction to normal samples taken after the mill has assumed a uniform condition.

(7) A method by which the data and curves obtained in one series of tests may be transposed and applied, as far as possible, for the purpose of securing further light on the phenomena disclosed by a second series.

ECONOMICS OF TUBE-MILLING

By S. J. Truscott

(April 13, 1912)

*Referring to the original paper on this topic by H. S. Ball, the table (see p. 199) registers the feed tests, among which Mr. Ball points to A as indicating the most efficient feed. The figures given under A, however, are not the actual figures of the test, but those after correction from the application of the moisture curve, whereby Mr. Ball would bring them from a condition of 33% moisture to one of 38%, at which the other tests were made. As I have pointed out, in making the correction for power Mr. Ball has made a serious mistake. The actual power consumed at 33% moisture is given as 5.3 hp. At 38% moisture, from the moisture power curve, it should certainly be less. However, Mr. Ball has increased it from 5.3 to 5.8 instead of decreasing it, noting with satisfaction that this figure conforms so well to the power-curve which he drew without its help. It will be seen that if this power had been logically corrected it would, on the contrary, have destroyed that curve.

It may be argued that in any case the results of test A, as they actually stand and without the advantage of the correction for moisture, still indicate the most efficient feed, the relative mechanical efficiency of that test being 7.76. By reference, however, to the table of crushing efficiency given in the original paper, it can be shown that that statement is open to very great doubt. In that table, which is derived from the feed tests, there is a column giving the number of tons crushed through 120-mesh per 24 hr., and it will be seen that for the experiment A, Mr. Ball gives the highest figure of 3.96 tons. But this is a figure which can be shown to be entirely wrong. All the other results in that column were obtained directly from the actual figures of the different tests. With that for A, however, this was not the case. This was obtained by measurement from the curve where, between the points O and R, Mr. Ball has not ventured to put in the actual results for A, but has drawn a freehand curve which places A as high as he though for his efficiency curve it ought to be.

Using the figures given by Mr. Ball, it can be directly calculated that the amount of material which would be crushed through 120-mesh, in that experiment, in 24 hr., is 2.79 tons, or, if it be increased to the greater efficiency of 38% moisture, 2.97 tons. With these figures—and they are the only ones which can be justified from the data given—the crushing efficiency of experiment A is far from being the best of the lot. It may be argued that crushing efficiency is not a measure of mechanical efficiency, since it does not take the power into account, but it is in any case an effective measure of the work done, and Mr. Ball in more places

*From the *Bulletin* of the Institution of Mining and Metallurgy. A discussion of the article by H. Standish Ball, *Mining and Scientific Press*, September 23, 1911. (Reprinted page 196 of this volume.)

than one has not failed to recognize the correctness of the indication it affords as to efficiency. The relatively low crushing efficiency of this test A, therefore, subjects to considerable doubt Mr. Ball's conclusion that his results show a critical feed in the experiment A. In this connection I take the word "critical" to mean a position from which a small departure will cause a serious change of efficiency. To show the difficulty, however, of checking Mr. Ball's figures here and elsewhere throughout the paper, I would refer to the number of tons for the experiment C in this table of crushing efficiency. This is given as 2.54, whereas in the general summary of tests near the end of the paper it appears as 2.34, and if it be worked out from the data on page 22 the figure 2.68 is obtained.

After the tests on feed come those for moisture. As Mr. Ball remarks, the outstanding feature of these tests is the great efficiency of a moisture of 37.7% above all others, this being the percentage of moisture in test B. It is seen, however, that this efficiency is due entirely to the lower power recorded. Further inspection shows this low power to be so singular as to require confirmation. This view is strengthened by reference to the moisture diagram. The black dots with circles around them in that diagram represent the power readings, and it is seen how entirely unsupported the position of B is, and how entirely isolated. So much is this the case, that there are, I think, few who would be bold enough to include it in drawing any power-curve. Mr. Ball appears to take comfort and confirmation from the fact, which he considers notable, that the power-curve is practically the efficiency curve inverted. But the fact that when the power is given low the efficiency appears high, is a natural and not a notable fact. He would again fill the gaps in his own experiment by bringing other authorities to his help.

Walter Neal, who from his experiments concluded that there was a definite critical point in the percentage of moisture, is the first one named. It was he, indeed, who first used the expression "critical" in this connection; but, if his results be carefully examined, it will be found that out of his eight experiments the one made in the neighborhood of 39% moisture had the advantage of the others in the largest and coarsest feed, two conditions which might well of themselves account for the greater efficiency at that moisture. I do not, however, wish it to be thought that I consider that Neal's experiments are thereby completely vitiated. I believe that they point to a greater efficiency in the neighborhood of 39%, but I do not think that they indicate any critical position. Sherrod is quoted as considering 40 to 45% moisture as the best under the circumstances of equipment, thereby indicating that in his opinion a range of good efficiency existed.

Of Smart, it is stated that in the tests carried out upon the Rand he found 38.5% as the most efficient, but on examining the reference to this authority, it is seen that when speaking of the introduction of Caldecott cones at a plant in Mexico, he said: "As

under the conditions of this plant 38% moisture gives a considerably higher tube-mill efficiency than higher or lower dilutions, a satisfactory tube-mill product was in this way secured." He was not speaking of any particular moisture tests made, certainly not of any made upon the Rand.

Lastly, there are the experiments of Fox, of whom Mr. Ball says that he "discovered that, taking the power consumed in conjunction with the fine grinding, the most efficient moisture would be 39.6." On examining the reference to that authority, it is seen that Fox says: "Comparing Fig. 3 and 4, it would seem that after the percentage of solution in the pulp has reached about 35%, both the fine grinding and the power consumption stay reasonably uniform," a statement which shows that he was quite unaware that his experiments had indicated a critical moisture, at 39.6. These two figures are those given in Fig. 10 on page 46 of Mr. Ball's paper, but in transcribing the curve marked 'moisture efficiency,' a serious alteration in its properties has crept in, which places the highest efficiency at the moisture of 39.6, whereas the real curve shows that an equally high efficiency was obtained at about 35% moisture, this being the percentage which Fox himself selects for comment. It is further to be noted that if the power be taken into account, the highest mechancial efficiency in these experiments by Fox occurs with a moisture of 28.57 per cent.

Concerning the pebble load, three tests were made which appear to indicate that a light load is mechanically more efficient, though a heavier load crushes more, these results agreeing with those made by Fox.

Concerning the speed, four tests were made, from which it would appear that Mr. Ball's tests confirm general experience. A remarkable and somewhat doubtful result is that which shows that it might take considerably more power to drive a charged mill at a certain lower speed than a higher one, Mr. Ball's experiments having shown this to have been the case as between 33 and 37 revolutions per minute. It is true that toward the end of the paper Mr. Ball gives results which would indicate a lubricating effect of the feed. Such internal lubrication of the feed can be conceived, but speed as a factor of lubrication is decidedly more difficult to grasp, and until further confirmation of such a property of speed is available, it would appear more easy to conceive that the power readings may have been wrong, a possibility mentioned by Mr. James. There is, in fact, some reason to doubt the power readings throughout. In addition to the considerations just given, I have already mentioned the singularly low and doubtful power in the case of experiment B of the moisture tests. It is further pertinent to remark that the power consumed in the condition of 'running light' throughout the experiments varies also very considerably. It is stated that the "motor was first run light for a few minutes to enable the power consumed by the shafting, belting, etc., to be ascertained." Seeing that this is a carefully specified duty, it is pertinent to ask whence come, then, the variations

from 2733 watts to 8640 watts in the power consumed for this purpose.

Finally, Mr. Ball desires to test his work by first calculating what the efficiency under certain set conditions would be, and then running an actual test under those conditions. He wishes to know what the efficiency would be at 7.2 tons feed, 35% moisture, pebble load of half volume, that is, 1200 lb., and 37 r.p.m. By calculation he finds this efficiency to be 4.27. He then runs an experiment, designated U, to test this result, and an efficiency of 4.32 is obtained. These figures agree well, but unfortunately the test U was carried out at 41 r.p.m. If it be argued that in stating the problem the figure of 37 was by mistake given in the place of 41, and that this was obvious from Mr. Ball's calculation on page 50, that explanation may be accepted. But in such a case the problem represents, with only one variable, the conditions under which the test P was run, this variable being moisture, and the efficiency given to that test in the statement on page 13 has only to be corrected for moisture. This correction should therefore be carried out on exactly the same lines as the correction for the experiment A previously considered, the two positions being as follows:

Experiment A.

Actual Conditions

Feed	18.6 tons
Moisture	33%
Pebble load	1200 lb.
Rev. per min.	41
Efficiency	7.76

Required the efficiency at 38% moisture.

Experiment B.

Actual Conditions.

Feed	7.2 tons
Moisture	38%
Pebble load	1200 lb.
Rev. per min.	41
Efficiency	4.45

Required the efficiency at 35% moisture.

It will be seen, however, when comparing the calculations, that the two were not conducted upon the same lines. In both cases reference was first made to the moisture curve, and a difference between the two moisture efficiences was obtained. On page 14 of his paper this difference was treated proportionally to figures of tonnage, whereas on page 50 it was treated proportionally to figures of efficiency. One method or the other must therefore be wrong.

In conclusion, Mr. Ball made an experiment under conditions all four of which represented, according to him, critical conditions, and he obtained a result which, he says, surpassed expectations. It would have been thought that if Mr. Ball had had a clear con-

ception of the problem in hand he would have essayed to calculate the possible result, and that he has not attempted to do so weakens the force of his argument considerably; there is little doubt also that had he done so his calculated results would have been very much higher than actually obtained. This final test gave an efficiency of 8.82, a figure the significance of which can be readily challenged. On page 54 the figures of the actual test are given, from which it can be calculated that the crushing efficiency developed under those four critical conditions was at the rate of 2.97 tons per 24 hr. Comparing this efficiency with those given on page 14, it is seen that, neglecting that of A, which is incorrect, it is surpassed by two others, namely, O and R. If, therefore, any gain of relative mechanical efficiency has resulted, it has been brought about at the expense of a loss in crushing efficiency. There is another side from which this result of 8.82 may legitimately be challenged. On page 4 Mr. Ball states the general experience, in saying that "high efficiency could be obtained only by feeding a sand containing a minimum of slime." Referring to the gradings of the sand fed in these experiments will show that, whereas the sand of this last experiment contained only 12% of material passing 120 mesh, the average amount in the case of the others was 18.8%, and there was not one which came nearer than 16.5%. In other words, the material fed in the case of the last experiment was appreciably coarser than in all others, and the result of 8.28 was obtained, therefore, under exceptionally favorable circumstances outside of the four conditions of the test.

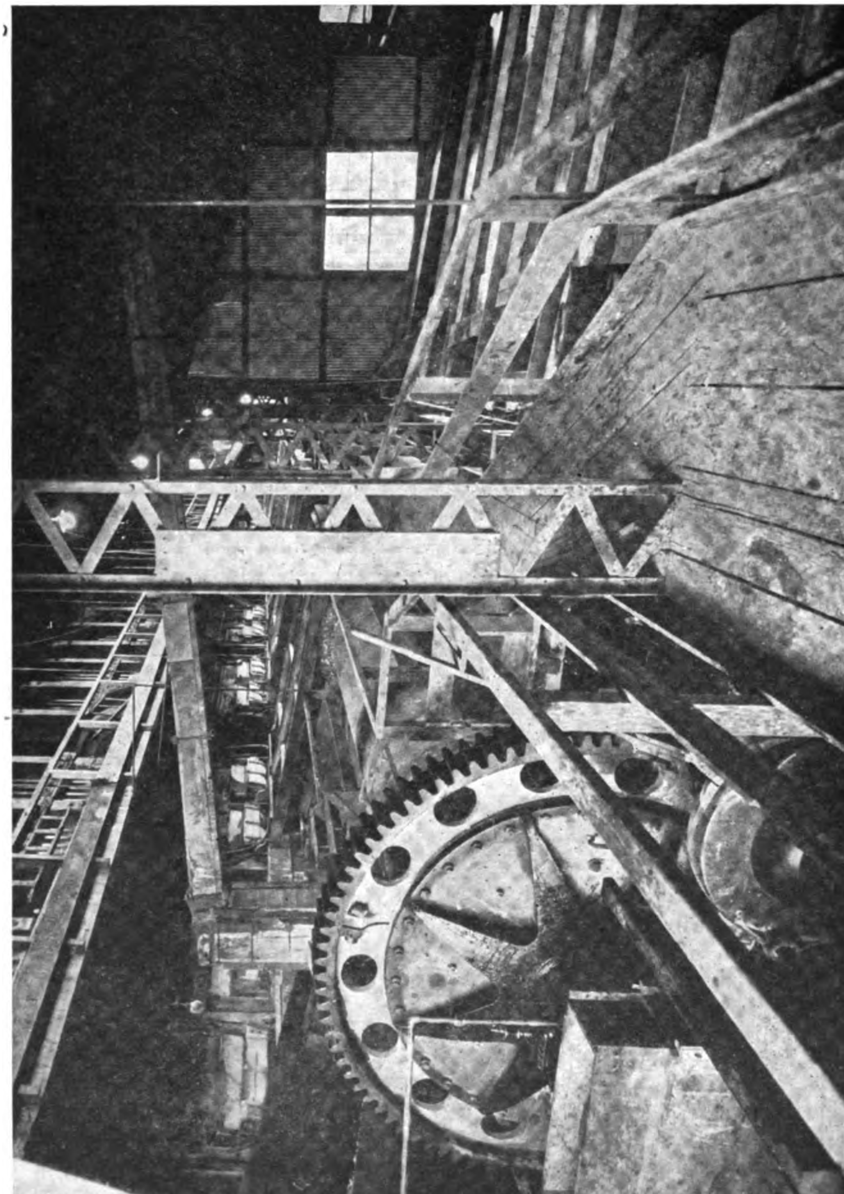
REGRINDING AT THE PITTSBURG SILVER PEAK MILL

By S. J. Kidder

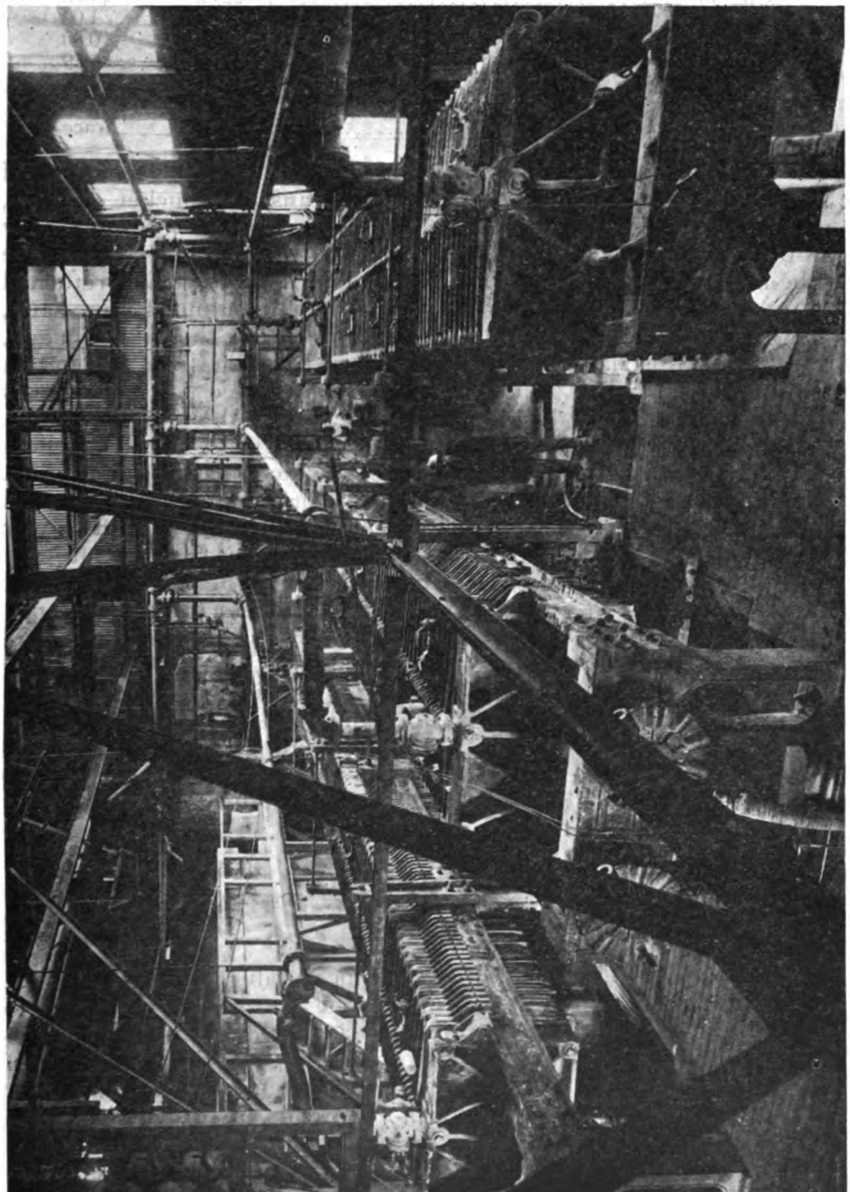
(February 23, 1913)

In the treatment of comparatively low-grade ores the rather high cost of all-sliming has in some instances made this procedure of doubtful value. On the other hand, there are numerous cases in which careful classification of the sand product, followed by the regrinding of the coarser sand particles, will result in a material saving above the cost of regrinding. The latter condition obtained at the Pittsburg Silver Peak mill, where crushing to 30-mesh with 120 stamps was followed by amalgamation, separation into sand and slime, direct treatment of the slime in Merrill slime-presses, and leaching of the sand.* Repeated screen analyses of the sand residue showed that the material remaining on 80-mesh, amounting to from 52 to 56% of the whole, contained 85 to 90% of the total value in the residue.

*For further details of the plant see *Mining and Scientific Press,* May 8, 1909, and page 632 this volume.



STAMP BATTERIES, TUBE-MILLS, AKINS CLASSIFIER, AND SECONDARY PLATES



MERRILL PRESSES AFTER ENLARGING TO 86 FRAMES, PITTSBURG SILVER PEAK

SCREEN ANALYSIS OF SAND RESIDUE

Mesh.	Per cent.	Value.
On 40	22.82	$1.83
60	29.32	1.00
80	10.05	0.80
100	10.47	0.59
150	7.75	0.28
200	10.72	0.18
Through 200	8.87	trace

By careful hydraulic classification, 50 to 60 tons per day of the coarsest sand is now returned, by means of a bucket elevator, using 6 by 10-in. buckets, to a 13-ft. Akins classifier feeding a 5 by 18-ft. Allis-Chalmers chain-driven tube-mill. The discharge of the tube-mill undergoes secondary amalgamation before being returned to the main classifying cones. As an aid in regrinding the fine sand, and at the same time with a view to increasing the mill tonnage, one battery discharges directly to the Akins classifier through a 4-mesh screen. The Akins classifier has worked for six months with no repairs other than increasing the size of the truss rods from ⅝ to ¾ in. The mill was originally lined with 4-in. silex, which lasted six months on the hard quartz which is being reground. Mine quartz, sorted out and sacked on the crusher-room floor, is used almost exclusively in place of pebbles. The quartz at the lower tramway terminal is spalled to the right size for feeding into the tube-mill spiral on the feed end of the mill. The quartz is much harder than that in either Tonapah or Goldfield ores and stands up well in the tube-mill. The screen analysis of the discharge pulp is practically the same as when using all Danish pebbles. For elevating the sand and for regrinding, 4 hp. is required, and 49.5 hp. for operating the Akins classifier and tube-mill. The total saving by regrinding the coarser sand amounts to from $1200 to $1600 per month. The additional milling cost on account of regrinding amounts to 3.9 cents per ton of ore milled, as follows:

Labor	$0.004
Supplies	0.015
Power	0.020
Total	$0.039

On account of the additional slime produced by regrinding, and an increasing amount of slime in the ore, it was necessary to increase the filtering capacity of the plant. This was done by enlarging the three 64-frame Merrill slime-presses by the addition of 22 frames and plates to each press. Arrangements were made for sluicing the enlarged presses from both ends instead of from one end as formerly. The cost of installing the 66 additional frames was considerably less than three-fourths of one 64-frame press. Each press was out of commission but from 24 to 32 hours in making the extensions. The only concrete required was for the three piers supporting the rear end of the presses, and no additions were required to the building.

The total operating costs for 1912, during which time 172,482 tons of ore was milled, was as follows:

	Labor.	Supplies.	Power.	Total.
Stamping	$0.067	$0.092	$0.155	$0.314
Amalgamating	0.039	0.004	0.001	0.044
Tube-milling	0.001	0.013	0.012	0.026
Neutralizing and settling	0.028	0.053	0.002	0.083
Leaching and sluicing	0.047	0.079	0.014	0.140
Filter-pressing	0.035	0.061	0.017	0.114
Precipitating	0.004	0.036	0.001	0.041
Refining	0.018	0.028	0.001	0.047
Assaying	0.016	0.012	0.003	0.031
Water service	0.024	0.016	0.027	0.067
Heating	.....	0.003	.....	0.003
Superintendents and foremen	0.067	.....	.....	0.067
Total direct operating	$0.346	$0.397	$0.234	$0.977
Pro-general	0.034	0.060	.....	0.094
Suspense account	.....	0.050	.....	0.050
Total operating, 1912	$0.380	$0.507	$0.234	$1.121
Total operating, 1911	$0.370	$0.477	$0.186	$1.040

The slight increase in operating costs in 1912 over 1911 is to be explained: (1) by an increase of 4.8c. per ton of ore milled in the contract price for power; (2) by a decrease of 10,000 tons in the amount of ore milled for the year, due largely to labor conditions in the summer months; and (3) by an increase in the suspense account of 5c. per ton, to pay for the improvements made during the year.

STANDARD SCREENS FOR GRADING ANALYSES

By H. Stadler

(October 19, 1912)

*After full and careful consideration and after ascertaining the views of kindred societies and of a large number of authorities in various parts of the world, the Institution of Mining and Metallurgy adopted, in November, 1907, a series of specially manufactured standard laboratory screens for the purpose of bringing about unification in the measurement of screen products. In order to meet metallurgical requirements it was necessary to put a large number of screens at the disposal of metallurgists, but as in ordinary practice only a comparatively small number of screens is required, it is suggested, with a view to securing still greater uniformity, that engineers and metallurgists, in making a choice, should preferably adopt the screens specified in the annexed table. The selection of the screens of the restricted set here recommended has been made in such a way that the apertures are in agreement

*Abstract from the *South African Mining Journal.*

with, or very close to, those of the mathematically correct reduction scale, so that the series of the products of these screens will show a regular reduction in volume, or weight, of 1 :4 from grade to grade.

STANDARD SCREENS FOR GRADING ANALYSES

Mathematically correct scale. Reduction ratio 1 :4. Ord. No. mech. val.	Mesh aperture.		Nearest I.M.M. Standard laboratory screens. Mesh per lin. in....	Mesh aperture.		Commercial Hand sieves. Mesh aperture, approximate......
EU.	In.	m/m.	No.	In.	m/m.	In.
0	1.0	25.4	..	....	.....	1
2	0.630	16.0	..	....	.....	⅝
4	0.3969	10.080	..	....	.....	⅜
6	0.250	6.35	..	....	.....	¼
8	0.1575	4.0003	..	....	.....	5/32
10	0.09922	2.52	5	0.10	2.540	
12	0.06250	1.5874	8	0.062	1.574	
14	0.03937	1.0	12	0.0416	1.056	Grading
16	0.0248	0.630	20	0.025	0.635	by
18	0.01562	0.3968	30	0.0166	0.421	classification
20	0.00984	0.250	50	0.01	0.254	(Quartz) Velocity m/m per sec.
22	0.00620	0.1575	80	0.0062	0.157	16.0
24	0.00391	0.0992	120	0.0042	0.107	6.35
26	0.00246	0.0625	200	0.0025	0.063	2.52
28	0.00155	0.0394	..	....	.....	1.0
30	0.00098	0.0248	..	....	.....	0.4

The diameter of wire of the Institution of Mining and Metallurgy standards is equal to, or very closely approaches that of the clear mesh aperture. The area of discharge is therefore fairly constant for all screens. The 50 and 80 I.M.M. mesh have the same mesh apertures as the 60 and 90-mesh screens, respectively, already in common use on the Rand. As the difficulties attending the manufacture of the 200 I.M.M. mesh have not yet been overcome, and as the commercial so-called 200-mesh screens are very unreliable, and in fact not much finer than the 120 I.M.M. mesh, it is more advisable to drop for the time being the 200-mesh altogether and to rely on classification for grades beyond the 120 I.M.M., which with an aperture of 0.0042 in. marks actually the limit of accuracy for screen measurements. When screens are described simply by the mesh number it will be understood that the I.M.M. standard is referred to. In reporting grading tests it is desirable to state whether wet or dry screening has been employed.

MUNTZ METAL PLATES

(Editorial July 1, 1911.)

The valuable description of milling practice at the Komata Reefs mill which Mr. S. D. McMiken presents in this issue derives added interest from so closely following the description by Mr. C. A. Chase of the work at the Liberty Bell. Comparisons are always useful, and the divergent results obtained by the use of muntz metal plates at the two mills is an interesting minor matter. It may be urged that in the one mill crushing is carried on in cyanide solution, while water is used at the other; but there is no known reason why cyanide should have any such effect. Muntz metal is an intermetallic compound of copper and zinc. Both mercury and gold are soluble in copper, but neither is soluble in the copper-zinc compound; therefore, as Mr. McMiken observes, absorption of amalgam by the plates is very slight. Possibly the hardness of the plates, found objectionable at the Liberty Bell, is due to the fact that muntz metal cannot hold so thick a film as a copper plate, an objection that is accentuated by the lack of absorption of gold into the plate. By far the best plate is one which is made of silvered copper, not scraped too closely during the clean-up. Muntz metal plates were first used in the Hauraki district because copper plates were not to be obtained locally, while muntz metal plates, intended for ships sheathing, could be had in abundance. Though used successfully in New Zealand, they have found but little favor elsewhere. The cyanide process was first successfully applied at the nearby Crown Mines at Karangahake, and Mr. McMiken's description of the methods of work at the natal place of the Pachuca tank is full of interest.

CONCENTRATION AND TREATMENT OF CONCENTRATES

CONCENTRATION OF SLIME

By Edwin A. Sperry

(August 6, 13; October 1, 1910)

INTRODUCTION

In considering the treatment of slime by concentration, there are many things which come up for discussion and comparison. Such a discussion necessarily commences with the methods employed in the work of crushing the ore and passes along through the various steps of the operation of preparation, to the final treatment of the slimed ore and the recovery of the metals in a marketable form. The subject will be discussed under five heads in their natural sequence. These heads are: (1) crushing and grinding; (2) sizing; (3) classification; (4) dewatering; (5) final treatment.

Crushing and Grinding

While the word 'crushing' is employed in this heading, it will not be used in reference to coarse-crushing machines, such as rock breakers, but will be applied to those employed in reducing the ore to less than 30-mesh, 30 wire (0.02-in. or 0.5-mm. open space). Sizes coarser than this come under coarse concentration and are, therefore, outside the scope of this paper. A definite classification of the various sizes to accord with the different lines of treatment which can be applied, based on practical lines, may be made as follows:

1. **Coarse,** or that size which can ordinarily be treated by jigs and including from the coarsest down to 30-mesh, 30 wire (0.02 in 0.5 mm. open space).

2. **Sand,** or that size which, passing 30-mesh, remains on 100-mesh, 38 wire (0.004 in. or 0.1 mm. open space). This size is readily treated on the common form of concentrator such as the Card or Wilfley.

3. **Fine,** or that size which, passing 100-mesh remains on 200-mesh, 45 wire (0.002 in. or 0.05 mm. open space). This size is readily treated on the Frue vanner, Wilfley slime table, Sperry slime table, and various other machines used for this purpose. The principal reason, however, for making this division is based on the speed of settling in a body of water. Ore particles between 100 and 200-mesh have an appreciable ponderosity and on this account will settle readily, and are easily separated from the finer size and easily treated.

4. **Slime,** or the very finest size. The term 'slime' means widely different things to different people. In previous discussions three distinct points of view have appeared: (1) The purely practical, according to which slime is considered to be that portion of the ore that is so fine as to make it unfit for treatment on the common forms of concentrators; this is ordinarily placed at minus 100 mesh, and the division is based on a dividing line between the

different methods of treatment. (2) The second point of view occupies an intermediate position between the practical and technical. The definition here might be stated as including that portion of the ore which, on account of its fineness, requires considerable time to settle in a body of water. This, if stated in terms of mesh, might more closely approach 200, but this is merely relative and given as a matter of comparison. This definition is practical in that it recognizes a division between quickly and slowly settling particles, and technical in so far as it recognizes a condition beyond terms of mesh in which specific gravity plays but little part. (3) The last and purely technical, is one according to which the particles are of such fineness that they lose their individual integrity and, in combination with water, approach, if they do not merge into, the colloidal condition of a hydrate, which changes their form and nature.

In the following discussion, the intermediate position will be assumed as more nearly conforming to the conditions to be met in practice.

In the crushing of ore, one of two results is aimed at, one being the least possible production of slime, the other, the complete reduction of the ore to this condition. In the discussion of the various types of machines for crushing and grinding, the crushing action, as well as the machines mentioned as examples, will be given as nearly as possible, in the order in which they theoretically occur, beginning with those which would give the least proportion of slime, and ending with those which would be most efficient in the reduction of the entire amount to slime. They will be first discussed from as nearly a theoretical standpoint as possible, analyzing the action of the machines themselves, and they will be placed, as nearly as possible, in their natural order as regards practical results.

In theoretical order, it happens, the Huntington mill stands ahead of the Chilean mill but, owing to the inferior method of screening in the Huntington, the order will be reversed. This will be more fully dealt with later.

The subject of crushing will be first taken up under the separate heads noted below. The different crushing actions may be classified as follows:

Character of Crushing Action Relative to Theoretical Proportion of Slime Produced

Elemental crushing action.	Derived actions.
1. Projectile.	1. Pressure-impact.
2. Pressure.	2. Pressure-torsion.
3. Impact.	3. Impact-torsion.
4. Torsion.	4. Impact-grinding.
5. Grinding.	

These may be placed in order of theoretical production of slime, based merely on the action itself, but subject to change from modifying conditions arising from individual action of the machine as follows: (1) projectile—Marx mill; (2) pressure rolls; (3) impact—stamps (California practice); (4) pressure-impact—Huntington, Wild; (5) pressure-torsion—Chilean mill, triplex

rolls; (6) impact-torsion—stamps (Gilpin county practice); (7) (torsion—disc-grinders; (8) impact-grinding—tube-mills, ball-mills; (9) grinding—cone grinders.

(1) **Projectile.**—This action is ideal for the disintegration of the ore as the resulting impact against the 'anvil' has simply a shattering effect. The shattering is naturally along the lines of cleavage and is nearly in conformity with crystalization. Several attempts have been made to adapt this principle but no thoroughly successful machine has yet been evolved. One machine of this type, however, has been recently presented which gives promise of good results—the Marx pulverizer. It is still experimental but shows points of interest.

(2) **Pressure.**—Rolls, in ordinary practice, exemplify this action as nearly as possible and stand at the head of the machines of this class. By the term 'ordinary practice' is meant the common type of rolls in contradistinction to high-speed rolls. These were run at so high a peripheral speed as to approach the action of impact rather than of pressure. Triplex rolls are placed in another class as will be explained later.

(3) **Impact.**—As rolls are an example of pressure action, so the stamps stand for impact. This contemplates the simple reciprocating motion without the rotating motion which is sometimes given them, especially in Gilpin county practice. This will be spoken of later. A distinction is made between California and Gilpin county methods. In the California mill stamps weigh approximately 1000 lb., drop 100 per minute, with height of drop 7 in., height of issue 2 to 4 in.* With these dimensions and speeds the delivery of pulp from the battery is rapid and as a consequence the removal of undersized particles is more thorough. Even with this rapid delivery there is much of the undersize which remains in the crushing zone but, owing to the rapid drop and consequently excessive splash, these will not readily remain on the die to be struck again. In feeding the stamps the proportion of slime can be influenced. If the feeding is high the ore is crushed against itself and is not so finely pulverized as it is when 'iron strikes iron,' as the expression is. Then, too, if plenty of water is used the undersized particles are the more readily washed out through the screen. By careful manipulation stamps can be made to crush very uniformly, with a small proportion of slime.

(4) **Pressure-Impact.**—In the Huntington, the Griffin, and the Wild mills, we have examples of this action. In these machines crushing is done on the inside of a large ring by the pressure, generated by centrifugal force; a muller in the form of a small cylinder rolling against the ring. In one form, the Huntington and Griffin, the ring and muller lie in a horizontal plane and the pressure is generated by centrifugal force. In the other form, the Wild, the ring and muller are in a vertical plane and the pressure is that of gravity. The compound action of pressure-impact is ac-

*The Gilpin county data will be given later.

corded these machines as the rolling-pressure action of the mullers is often combined with jumping of the muller, the return of which to contact with the ring develops considerable impact. This is due to the fact that the mullers are not rigid and are free to recede. In the first form the ore, after crushing, is thrown vertically and then, by vanes, horizontally to the screens. In the Wild it is thrown, at once, horizontally, to the screen. In comparing the two actions the difference in the efficiency of these mills, due to these causes, will be appreciated by millmen, the Wild mill conforming more nearly, if not quite, to the theoretical.

(5) **Pressure-Torsion.**—This action is shown in the Chilean mill and in the Triplex rolls. It is the result of rolling a short cylindrical-shaped body in a circular direction on the face of the disc or flat ring. As the muller is a cylinder, and not a cone, its line of contact with the horizontal plate is equidistant from its axis, and as this contact line is always radial to the disc, or ring, there is a constant twisting motion between the two surfaces. The weight of the rolling member in the Chilean, and the pressure of the spring in the Triplex rolls, with the torsional or twisting action described, give the 'pressure-torsion' combination. While these two machines are theoretically in the same class, the lower screening capacity of the Chilean places it below the Triplex in the practical order. The pure torsional action would be more highly productive of slime but as it is slight in both cases mentioned, it does not enter largely into the consideration of results. The Triplex rolls have the advantage of an immediate removal of the ore from the crushing zone and, as in the case of ordinary rolls, the undersize is isolated at once, nothing but the oversize being returned to the crushing zone, at least, such should be the case.

(6) **Impact-Torsion.**—Gilpin county stamps are used as an example of machine using this combination. In comparison with the California stamp the general form will be given. The stamps weigh 650 lb., drop 30 per minute, from a height of 17 in., with height of issue 10 to 15 in. Owing to the height and upward slowness of the motion of the stamp, the cam has an opportunity to give the stamp a spinning or whirling motion, and in descending it strikes the die with a twist. This gives the combination of 'impact-torsion.' The torsion is not great, but more than in the Chilean mill. The slow action of the stamp, together with the great depth of the issue, causes fine grinding which places it far down in the practical scale.

(7) **Torsion.**—There are no machines which use a strictly torsional action in crushing ore, the nearest approach to it being the disc grinder, composed of two discs revolving in opposite directions. The ore is fed at the centre between them. The discs are slightly dished in shape, that portion near the edge having parallel faces. They are so arranged that these faces are a certain distance apart and are set up when the wear increases the space. The circular motion of the discs gives a twisting motion which is most

effective in crushing. This form of machine is not much used as the wear on the crushing faces is excessive and the power consumed is relatively large.

(8) **Impact Grinding.**—The tube or pebble mill, is used as an example of the application of this combination. The general form is a tube, ordinarily 14 ft. long by 5 ft. diam.; lined to protect the outer shell of the mill, usually with silica brick, or some such material. The tube is charged with several tons of rounded flint pebbles of 3 to 4 in. diam. The ore is fed at one end and the tube revolved. As the pebbles roll and fall down from the ascending side, the ore is caught between them and eventually ground into a slime, when it is discharged from the other end. The object of this machine is to reduce everything to slime and it is considered one of the most effective devices for this purpose. The ball-mill is a modification of the tube-mill with the substitution of steel balls for pebbles and steel lining in place of the silex brick. It is designed for dry grinding and is used extensively in cement manufacture.

(9) **Grinding.**—This action is exemplified in the ordinary sample or cone grinder. Its action differs from disc grinders in that the moving surface passes the fixed plates in practically straight lines, causing a simple rolling motion without torsion. It is so designed that by adjustment, the grinding surfaces gradually approach each other until they are in contact. This machine has no practical value in metallurgical operations outside of the laboratory.

With this short discussion of the various actions the comparisons of practical results can be more readily understood. It is an extremely difficult matter to make any specific statement as to what machine will produce the greatest amount of slime, for, to determine this point, it would be necessary to operate all of them on the same ore and under the same conditions. The only way to reach a conclusion is to analyze the crushing action and then, by a consideration of the manner in which the ore is delivered from the crushing zone, as regards its promptness, a fair idea can be gained as to its efficiency. One statement can be made which needs no argument, and that is, that it matters not what machine or what action is employed in the crushing of ore, the amount of slime produced is largely, if not entirely, influenced by the promptness and thoroughness of the removal of the undersized material from the crushing or grinding zone. From this point of view, and from this only, can proper comparison be made between machines of different type. Based on this statement, it can be readily assumed that rolls would produce less slime than stamps, even of the California type, and the Chilean mill less than the Huntington. As to the stamps and the Chilean, it would depend on the manner of operating the stamps. With proper use the stamps produce less slime. While the Wild mill uses the same crushing action as the Huntington, it can be placed ahead of stamps, owing to its prompt delivery, and it would be fair to suppose that the Triplex rolls would be superior to the stamps.

A general statement concerning the machines cited can be made, however, by placing them in one of three classes: (1) those best adapted to crushing without producing slime; (2) those which occupy an intermediate position and can be used for either purpose according to conditions; (3) those especially adapted to the reduction of all the ore, or practically all, to slime. They may be classified according to this plan, as follows:

1.	2	3.
1. Rolls.	4. Stamps.	7. Disc-grinder.
2. Triplex rolls.	5. Chilean mill.	8. Tube-mill.
3. Wild mill.	6. Huntington mill.	

Many styles of crushing machines might be cited, but only those more commonly known have been used in comparison, it being more especially the present object, to define the various actions and results by reference to well known forms. From these it is hoped that analysis of action and results may be readily applied to such machines as may be met in special cases.

Sizing

Sizing is the separation of ore particles into classes having the same diameter, within a certain range, regardless of their specific gravity. Sizes are best specified by using two screen meshes. For instance, if it is stated that certain particles are 30-60 mesh, it is to be understood that they pass a 30 and remain on a 60-mesh screen. It is customary to designate a screen as '30-mesh' or '60-mesh' when from centre to centre of wires it is $^1/_{30}$ or $^1/_{60}$ of an inch. In other words there are 30 or 60 meshes to a linear inch. This term is indeterminate as the size of the wire has to be considered. In stating the size of a screen in terms of mesh it is always wise to designate the size of the open space and size of wire, as these are vital points. There is a method of designating screen sizes, which is coming into vogue, and by its use there is no chance for a doubt as to the space. It is the use of metric terms in designating the size of the open space direct. It should be encouraged, and were it not for the unfamiliarity of the public with the comparative value of the metric measurements, it would be more universally adopted.

There is no department in metallurgical work where there is so much confusion and misunderstanding as in the designation of sizes of screens. A great deal has been written on the subject, but there seems to be still room for some definite system of terms. The Institution of Mining and Metallurgy has put out a standard list, and for some few purposes it is commendable. In the presentation it stated that the design was simply for use in laboratory work in order that results could be based on some uniform system of sizes. It was based on a standard size of wire for each mesh, but it is obvious that this standard could not be carried into practical mill work and this fact was recognized by the Institute. The reason for this can be briefly stated. There is work in which heavy

service is required of screens having the same open space as there should be in others which require as high a percentage of open area as possible. As an illustration the screen of a trommel needs to be heavy to insure a reasonable length of life. In the jig cell handling the oversize from this trommel the open area should be as great as possible and on this account lighter wire can and should be used. Other cases might be cited, but this will illustrate the point.

In the average mind the term '30-mesh,' or whatever it might be designated, has a specific value and all that is necessary is to order or specify by this simple and ambiguous method. At many of the larger mills this confusion does not exist, as all the screen lines are worked out on definite statements of 'open space.' Even then, there has been but little information to guide the designer or operator in the selection of screens which have the required weight and space for the purposes for which it is intended, in order to fulfill the conditions of the service it is designed to meet. There is no greater need of care in any department of milling than in the

OPEN SPACE	64ths in.	25/64	23/64	20/64	18/64	16/64	15/64	14/64	13/64	11/64	10/64	9/64	8/64	6/64	5/64	4/64	2/64	1/64	1/128	1/256	1/512
	M.M.	10	9	8	7	6.5	6	5.5	5	4.5	4	3.5	3	2.5	2	1.5	1	0.5	0.25	0.1	0.05
	DECIM. IN.	0.394	0.354	0.315	0.275	0.256	0.236	0.216	0.197	0.177	0.157	0.138	0.118	0.098	0.079	0.059	0.039	0.02	0.009	0.004	0.002
MESH/WIRE		2/12	2/10	2/7	2.5/10	2.5/9	2.5/8	3/11	3/10	3.5/12	4/13	4.5/14	4.5/12	6/15	7/16	9/17	14/21	24/25	40/28	60/30	100/35
MESH/WIRE					3/17	3/15	3/13	3.5/15	3.5/13	4/15	4.5/16	5/16	5/14	7/18	8/18	10/18	16/23	28/27	45/30	80/35	150/40
MESH/WIRE									4/17	4.5/18	5/19	6/22	6/18	8/22	9/21	12/23	18/27	30/30	50/31	100/38	200/45
MESH/WIRE															10/24	14/30	20/32	35/35	60/35	150/47	

WIRE SIZES REFERRED TO U.S. STANDARD LEGAL GAUGE

selection of a screen line, so designed that the particles may be sized within the most desirably range. A failure to select the most advantageous sizes and ranges has contributed, in many cases, most seriously to failure. A fixed system of determining what open space is most efficient would be of great value to the designer as well as to the operator.

A table has been prepared which is herewith presented, in which the open space is given in sixty-fourths of an inch, in millimetres, and in decimals of an inch, starting from what is termed '2-mesh' and ending with '200-mesh,' under each head, giving size of open space. In one column several mesh values are given with the gauge number of the wire which would give the open space at the head of the column. For instance, in the column headed $^2/_{64}$—1—0.039, the open space would be $^2/_{64}$ in., 1 mm. or 0.039 in. Running down the column it appears that 14-mesh with No. 21 wire, 16-mesh with No. 23 wire, 18-mesh with No. 27 wire, or 20-mesh with No. 32 wire would all give the open space designated at the head.

With this table at hand, a line of screens may be readily and quickly worked out and the weight of the wire selected to meet

the various services required. The wire in the larger sizes is calculated to $^{8}/_{1000}$ of an inch, which is as close as it could be approached in ordinary application or manufacture. In the finer meshes it is practically correct. Inasmuch as there is considerable of a variation between the several gauges it would be well to establish a fixed standard in this regard. There is but little difference between the English legal and the American legal standard gauge, and either could be used without materially affecting the results.

From the nature of the case, sizing is necessarily done by screens. It is only within the last few years that screening has been successful for sizes finer than 30-mesh, 30-wire. The shaking or impact screen has made it possible to screen successfully as fine as 100-mesh, 38-wire. Of this type are the Impact, Imperial, Wild, and Centripact screens. The Impact employs a simple upward 'bump' at right angles to the screen, which, as the screen is placed on a slope, gives the ore a toss and lands it a little lower on the screening surface. The Imperial has a combination of horizontal and vertical motions, and as the ore is tossed, the screen recedes horizontally beneath it so that the ore lands at a point lower down on the screening surface. The Wild has a swinging motion without a bump, and advances the ore by a series of tosses. On the Centripact screen the ore progresses from the centre to the edge of a circular screen by a series of tips imported by a revolving ratchet. They are all open to a similar objection—the necessity of many wearing parts.

Another type of screen has been introduced which bids fair to take a prominent position. It might be called a washing screen, as the pulp is distributed over the surface of a moving screen and moved forward, to and under a spray which washes the undersize down through the screen, leaving the oversize to be discharged at the tail end of the machine. No agitation is needed in this type of machine, dependence being placed on the spray to remove the undersize. There are two forms in which this type has appeared, the first being that of an endless belt of screen-cloth having a slow, progressive motion, as in the Callow screen; the other being virtually an inverted trommel, screening from 'out to in' instead of from 'in to out,' as in the King screen.

J. M. Callow, of Salt Lake, thus describes the Callow screen: "The fundamental principle of this machine is a traveling belt of screen-cloth, over which the ore and its carrying water is spread by means of distributing aprons or feed soles. This belt of screen-cloth passes over the head and tail-rollers, and the belt is caused to travel continuously in a horizontal direction at a speed varying from 15 to 30 ft. per min., according to the nature and quantity of the material to be screened. These feed soles or aprons play a most important part in the operation of the machine, and its successful and proper operation depends in a great measure upon this feature, as well as on its proper construction and adjustment. It is with this that a preliminary sizing is done, for, as the particles, which are of all sorts and sizes, fall from the lip, the coarsest

pieces, having the greatest trajectory, strike the screen at a point far ahead of the smaller ones, leaving a space behind free and unincumbered, which permits the free passage of the fine material and water. Simultaneously with this action the cloth is moving forward and the deposit of oversize is continuously removed from the separating or screening zone, and thus the machine continues to perform its function so long as it is kept moving and is supplied with the feed. Beyond the screening zone, the deposit of oversize is carried forward and passes under a shaking spray of clear water, where any remaining traces of slime or of fine adhering particles are washed off and pass through with the water from the spray into the undersize hopper beneath. Continuing, the oversize still clinging to the screen-cloth, passes in front of a small impinging spray which is situated about the mid-diameter of the front roller, and the oversize is there washed off the screen-cloth into the oversize hopper below."

It is interesting to note some of the results accomplished by the use of the Callow screen, in actual mill work, as given by Mr. Callow. In one case, two screens 4 ft. by 24 in. were installed, screening to 40 and 100-mesh, respectively. The first screen (40 mesh) takes the undersize from a 6-mesh, 22 wire (0.137 in. or 3.5-mm. open space) trommel, and handles 150 tons per 24 hours, or 6.25 tons per inch of belt width. The life of the belt in this case is 35 days. The second screen (100-mesh) handles the undersize from the first screen, amounting to about 30 tons in 24 hours. The life of this belt is upward of three months. The oversize from the first screen goes to two jigs. Formerly three jigs were used to handle this product. The oversize from the 100-mesh goes to one Wilfley table, where four tables were formerly used, and the undersize from the 100-mesh goes to four Wilfley tables. The muddy water from these last Wilfley tables is settled, dewatered, and treated on vanners. The final results show a great reduction in value of the tailing. The jig tailing is reduced from 0.6 to 0.3%, and the Wilfley tailing from 1.3 to 0.35%. Several similar cases might be cited. This improvement in results is probably largely due to the increased thoroughness of the sizing and may be cited as a practical illustration of its great importance. The cleanness of the work has been shown in tests which have come to my notice, one being on an ore especially difficult to screen, which showed less than 4% of undersize remaining on the screen. In that the pulp is quiescent on the screen, the production of slime is slight in comparison with any screen in which agitation, shaking, or rolling is employed.

The action of the King screen is similar to the Callow, with the exception that a slight rolling motion is given to the ore. In cross-section it is hexagonal, the sides being curved opposite the circumscribing circle. A spray pipe is placed directly over the centre of the screen, on the outside, the purpose of which is to wash the pulp as it passes beneath it. The undersize material passes through the screen into its proper launder, while the oversize is

discharged over the edge into its hopper. There are sprays placed inside the screen below its centre which clean the screen of particles which may be lodged in the meshes. The slight rolling motion given the pulp seems to aid in the washing action without material injury to the screen. In one instance this screen was in constant operation for over four months without appreciable wear to the screen cloth. In this case, 100-mesh screen cloth was used on 30-mesh pulp, there being less than 4% of undersize remaining in the oversize. The same advantage in the matter of reduction of slime may be credited to the King screen as to the Callow, though in less degree, owing to the slight rolling motion already mentioned, but this can hardly be considered as material, as the action is slight, both in amplitude and time. The life of the screen appears to be somewhat longer than that of the Callow, the reason for which is probably in the fact that it is not subject to a continued bending as is the case in the Callow. The rolling action in the King may, however, offset this when comparisons are made in long continued operation. The capacities and efficiencies of the two forms seem to be nearly identical, and while the King screen has some advantage in the matter of weight and floor space, the Callow requires less height, which in some cases is of importance. In comparative work between the King screen and the trommel, and extending through several months, the proportion of slime produced in one mill, separating lead and zinc, was 20% for the trommels as against 12% for the King. These results were probably due, in part, at least, to the prompt and thorough isolation of the undersize by which only a slight proportion was returned to the crushing machines. It can be stated certainly, that in the development of this type of screen, as exemplified in both the Callow and King, a great advance has been made in the solution of the problem of sizing as regards 'fine' and 'slime' work, and several forms based on similar action have been recently introduced with satisfactory results.

Classification

Classification is sometimes confused with sizing, but they are based on diametrically opposite principles. A few comparisons will serve to more definitely fix the difference in mind. As previously stated sizing is the separation of ore particles into classes having the same diameter, within a certain range, regardless of their specific gravity. Classification is just reversed—it is the separation of ore particles into classes having the same settling velocity in water, within certain limits, regardless of their diameter. It is based entirely on the speed of settling which is directly influenced by specific gravity. An extended discussion on the sizes of equal-settling particles will not be attempted, but it will be sufficient to mention one comparison—quartz and galena. A particle of galena has about the same settling velocity as a particle of quartz approximately four times its size. From this it will be seen that there is a wide variation in the sizes that would be drawn off together. In concentration sizing is of great importance and with-

out it the treatment of the ore is rendered most difficult. As a result of classification the ore particles show extremely wide variations in the matter of size, as has been shown, far exceeding the range of sizes which, it would seem reasonable to believe, would give the best results. There is much discussion on this point, with honest advocates on both sides. One point held is that, as the quartz particle is so much larger than the galena, it extends farther up into the flow of water and is more readily rolled. If the action of carrying off the quartz particles were that of rolling this might be considered important, but as the ore is stratified before any elimination of quartz takes place, the latter must necessarily rest on the stratum of metallic particles. If, then, the quartz particle is rolled it is more than apt (and, in fact, this condition has been met and noted) to cut a channel in the mineral stratum and by this disturbing it afford the mineral an opportunity to pass off with the quartz. As a matter of fact, the quartz particles should move off as a body, gently, and in passing over the mineral, should not have sufficient weight to disturb the mineral stratum.

These arguments refer entirely to sizes larger than 100-mesh, 38 wire material, as the use of classification on sizes finer than this is not often advantageous. In these sizes the above objections cease to be serious. This is due to the fact that when these extremely small sizes are reached, the behavior of all the particles, either quartz or mineral, is similar. This is one of the reasons why the treatment of slime is difficult. While it has been stated as a fundamental fact that hydraulic-classification is theoretically imperfect, still it must be conceded that, beyond a certain point, the ability to size by screen is impracticable if not impossible. From this point, taken for the sake of argument at 100-mesh, 38 wire and finer, the practice must of necessity be hydraulic classification. As to the form of the appliance best fitted to do this work there is room for discussion. In the use of hydraulic classifiers in the form of the 'cone sizer,' as it is termed ,there is ordinarily excessive use of wash water. In this addition of water two serious results are brought about, one being the extra demand for water, and the other, the dilution of the pulp to such an extent as to cause trouble in its subsequent preparation for final treatment. In some cases the dilution of the pulp has extended to a 4% solid consistence. Now, when it is taken into consideration that slime pulp should be delivered to the table for final treatment at not less than 15% solid consistence, for some classes of machines, and as high as 25 or 30% for others, it can be readily seen how serious dilution may become. It has been stated, and is still maintained, that in some cases, sizing to 100-mesh is considered close enough in practical operations. There are cases, however, where there is a great advantage in isolating material which more nearly approaches the condition of slime, as indicated in the second definition given. One case which can be cited is that of a tailing which was being treated for the extraction of lead in an extremely fine state. It was found by experiment that practically all the recoverable mineral was ex-

tracted from material coarser than 200-mesh, 45 wire, and in order to obtain the best results the minus 200-mesh had to be isolated. For several reasons it would have been futile, if not disastrous, to attempt to make this separation by the use of the ordinary cone classifier. The additional water would have been prohibitive of good results in the final treatment, and it would have been impossible to divert a portion of the fine mineral particles into their proper channel as well as was done by the method employed.

Many forms of classifiers have been devised which, in most instances, can be placed in one of four types: (1) the cone-sizer; (2) the spitzkasten; (3) the spitzlutte; (4) the sizing-launder. In some cases it might be hard to typify a machine in which a combination of the several types is employed. There are two general methods of applying the principles on which hydraulic classification is based, one being by the use of a rising and the other by means of a horizontal current.

The first type, the cone-sizer, using a rising current and clear water jet, while being the one most commonly employed, has, in my judgment, the least to recommend it. The quantity of water necessary for its operation is excessive, adding materially to that already in the slime, which certainly should be avoided, while the liability of choking is great, and necessitates constant watching. A simple form of classifier is shown in the accompanying cut, Fig. 1. This classifier was designed to remove the muddy water from material being fed to a 10-mesh jig. The main idea was to change the flow from a rapidly increasing downward current to a uniform up-current, and at the same time to pass by, rather than to feed toward, the aperture leading to the draw-off. In this way the up-current would not have to overcome the down motion of the particle, and if its specific gravity were sufficient, it would overcome the rising current naturally and of its own weight.

As the pulp passes down at A, it increases its speed as it approaches the lower point of the classifier, and as it strikes the 'block' shown, it is deflected quickly into the channel, B, which is so designed that it has a uniform sectional area at all points along its entire length. As the pulp passes through the constricted area immediately above the 'block' it is thrown up into the uniform current for some distance, and if the speed of the current is not rapid enough to carry it along to the discharge it will fall onto the lower wall of the channel, B, and pass down through the ½-in. slot between the 'block' and this wall, into the discharging chamber below. In the meantime the water from the pressure pipe impinges on the bottom of the 'block,' and distributing, passes up quite uniformly through the ½-in. space and washes the muddy water from the draw-off material. It was found that by its use classification was excellent and the feed on the jig was clean. It is certainly cheap and simple, and a carpenter can make one easily in a day.

The second type, using a horizontal current, has much to recommend it. This type, as stated, is exemplified in the spitzkasten. In this form, not only is the excessive use of water avoided,

but much clear water is recovered, thus increasing the available water supply. Again, the pulp as drawn off is properly prepared

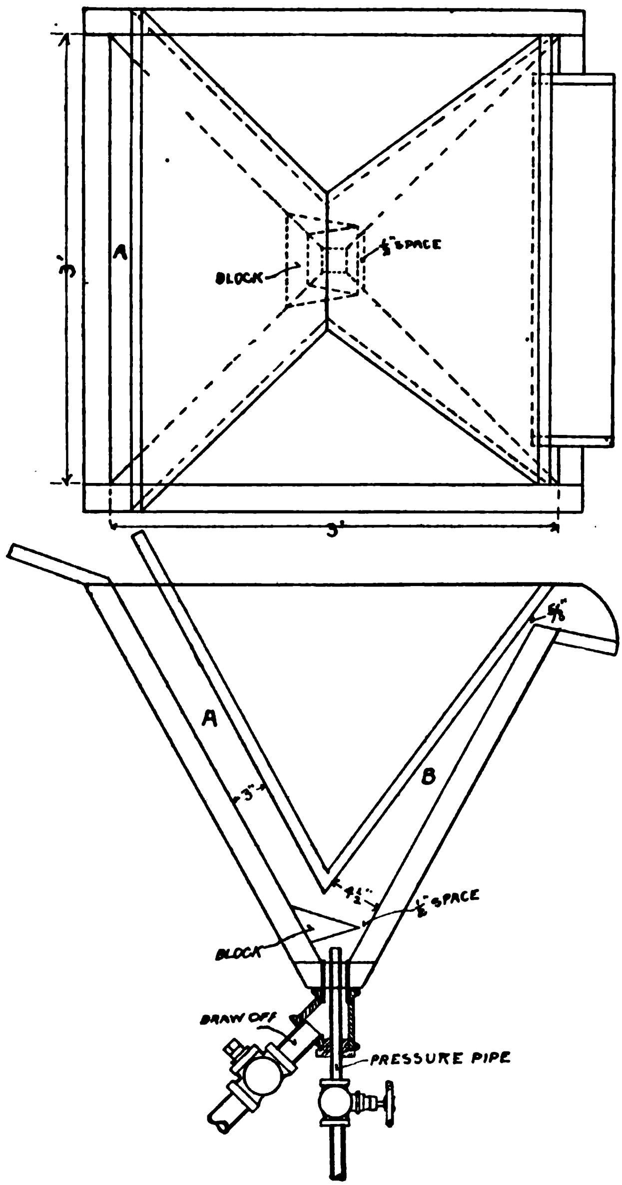


FIG. 1. CLASSIFIER

for treatment on the various machines designed for treating the different sizes, especially the slime. It cannot be denied that there is a tendency toward choking, but as the depth of water in the

spitzkasten is greater than in the cone-sizer, this can be more readily avoided. It can also be done by attaching a pressure or service pipe to the draw-off spout. By turning it on when necessary the clog can be broken. This pressure pipe is not so readily applied to the cone-sizer, as it is already employed in its regular operation. In one instance in my experience, which will be dealt with in detail under the head of 'Dewatering,' the classification in a spitzkasten, 8 by 10 by 30 ft., was almost perfectly done, grading regularly down in each of the four compartments, from 60-mesh in the first to 200-mesh in the last.

The third type, the spitzlutte, is made in the form of a V-shaped trough with the course of the flow transversely. A similarly-shaped, vertically-sliding trough is inserted in such a manner as to form an oblong channel bent at the middle. The sectional area of the channel is uniform at all points. This gives a down and an up flow at practically the same speed, the clear-water jet at the apex, compensating for the draw-off of the coarse material.

In the sizing-launder there is a resemblance to the spitzkasten much reduced in size. Several forms have come to my notice, some using enlarged sections with hoppers and clear-water jets, and some longitudinal slots in the bottom at intervals along its length. The simplest and most effective of those noted is the form constructed and used by Frank Bosco, in the Grand Mogul mill, at Gladstone, Colorado. Its normal form was in the shape of a V. At intervals the form was changed to that of a square-box launder for a distance of 2 ft. In this portion, a longitudinal slot was cut in the bottom along the centre line. The tendency of the coarse material was to collect at the apex of the V-shaped section, and as it debouched into the widened section this coarse material was delivered directly to the slot, and, passing down through it, was collected in a hopper attached to the launder and conveyed to its proper destination. Its operation and results were satisfactory.

Dewatering

While proper sizing is certainly an important step in the preparation of slime for treatment, settling and dewatering has probably more influence on the final result than any other step in the process. This is more readily and easily accomplished by the use of properly shaped settling tanks, more especially the spitzkasten. This appliance is a V-shaped tank, ordinarily 24 or 30 ft. long, 8 ft. high, and 10 fet. wide, the sides being brought together at the bottom. It is generally divided into sections by the use of sloping divisions usually 3 ft. high, so built as to form hoppers at the bottom. These sections are 6 ft. long and have draw-off holes at the apices. The flow from them is regulated by inserting plugs having holes bored through the centre and attached to long rods to be manipulated from above the tank. A series of plugs having different sized holes is made so that by the proper selection, the draw-off may be larger or smaller. A fuller description of the spitz-kasten will be given later. There is one principle in the

treatment of slime by concentration that is so often lost sight of by men who have been considered capable in their line. This is the matter of the absolute necessity of a thoroughly gentle method of coaxing, instead of a violent method of attempting to force the particles to do the will of the operator. The slime particle is a willful and elusive little fellow, and the old saying that 'you can catch more flies with sugar than with vinegar,' will certainly obtain in this case. It is better by far to let the particle seem to have his own way and then, after that way has been carefully studied and learned, to adopt such methods as will meet his wilfulness and set a trap for him into which he will naturally fall of his own accord.

The principle briefly stated is: in all operations connected with the handling of slime the utmost importance must be given to the avoidance of currents or agitation. The sooner this fundamental principle is fully understood and its full value appreciated, the sooner will the problem of the concentration of slime be simplified. This applies to all devices, as well as to the so-called improvements on otherwise efficient appliances in which, or by which, any kind of agitation, restriction of flow, disturbances, or currents of any kind are created. Among these can be classed the so-called 'baffle boards,' which are supposed to throw the particles down to the bottom of a settling device of any kind. Also, the form of spitzkasten which, being composed of a series of inverted pyramids, have the overflow from one to the other at or near the surface. Another is the 'over and under' settling tank containing partitions which are set alternately above and below through constricted sections.

There is another method of handling slime which I can freely say has little to recommend it. That is the use of the 'canvas plants.' I will assert, without fear of contravention, that, in comparison with the settling method, it is far inferior. In practical demonstrations, the saving of mineral has ranged from 40 to 50% for the canvas as against 60 to 80 for the settling method. This is borne out by actual test and experiment. In one particular case which came under my observation, there were 180 tons of ore crushed every 24 hours. Rolls were used. The mill tailing was screened on 20-mesh shaking screens. The undersize was further sized by the use of large settling tanks, the overflow from which was approximately 60 mesh or finer. This product amounted to 65 tons in 24 hours. Of this amount, 13.16 tons was caught on the canvas, and 51.84 tons was wasted. Taking lead as a basis of calculation, the average value of the 65 tons was 4.7%. The product from the canvas plant to be re-treated or cleaned was 20.2% of the total by weight, and averaged 12.1% Pb. containing 51.6% of the total lead. Of this lead 95% passed a 200-mesh screen, representing 49% of the total. The minus 200-mesh size of this product was 50% in quantity, and ran 23.2 Pb. The tailing or waste from the canvas plant was 79.8% of the total in weight, and ran 2.9 Pb., containing 48.4% of the total lead, 86.2% of which passed 200-mesh, representing 41.7% of total lead. The minus 200-mesh of this was

60.5% in weight and ran 4.1 Pb. It can be clearly seen that 90.7% of the total lead in the pulp fed to the canvas would pass 200-mesh, 45.9% of which was lost as tailing from the canvas plant. The product for re-treatment was subsequently run over a slime table, with a saving of 84%. It is fair to suppose that had the entire tailing been properly sized and prepared, at least 80% of the total lead could have been delivered to slime machines, and that on the same basis 67.2% of the total could have been recovered instead of 43.3, as was the case.

In direct comparison the case of another mill using a large spitzkasten for preparing the slime pulp may be cited. This mill was treating a somewhat similar ore, but crushing to 60-mesh. All tailing, amounting to 80 tons, was run into a spitzkasten 30 by 10 by 8 ft., having five compartments. The first four drawings were made for experiment, and to determine the value in the products. The average value of these four combined was 0.5% lead and the weight was 55 tons. The fifth draw-off, of 25 tons, ran 5% lead, and would all pass 200-mesh. The overflow water was perfectly clear, and while it must be admitted that some valuable slime was held in suspension, the amount was conspicuously small when compared with losses from canvas tables. Thus it will be seen that 82% of the value was retained in a product ready for treatment on the slime concentrators, while the product from the canvas plant contained but 51.6% of the value, and was not in proper condition for economical treatment, as it was necessary to rehandle it. After dewatering in the spitzkasten, a subsequent saving was made on the slime tables of about 80% of the lead in the pulp fed to them, or about 66% of the total lead.

Another point worthy of consideration is the possibility of re-using the clear water from the overflow of the spitzkasten for the purpose of washing in subsequent treatment, thus economizing in the use of water, which in many cases is important. Many other cases could be cited. They would be parallel in results and conclusions, so these are used as typical. In the use of the spitzkasten there is a series of operations carried on contemporaneously, all of which tend to the advancement of the work in hand. These operations are: sizing or classification, dewatering, and partial concentration. A sketch of one similar to that used at the mill referred to is shown. See Fig. 2. The sizing in this case was nearly perfect, ranging from 60-mesh to 200-mesh in regularly decreasing sizes, merging, of course, from one into the other. The ultimate object was, however, attained—the separation of the 200-mesh material which, of itself, carried such a large proportion of the escaping mineral. The consistence of the products from each draw-off was easily regulated by use of plugs and goose-necks, so that pulp was dewatered to any desired proportion of solids. The discharge was automatic, continuous, and regular, affording the best of conditions for proper subsequent treatment. It is obvious that by recovering the 200-mesh material in one product, since the bulk of the value is in this, a partial concentration was accomplished.

It has been stated that any current causes losses of slime. While this may be strictly true, there is a certain settling action in the slow movement of water in a body, which can be employed to advantage. This statement is based on observations made in several cases. For instance, a portion of the pulp at one mill was placed in a closed bottle and shaken thoroughly. At the end of 12 hours the pulp had but imperfectly settled, the water still being cloudy. The pulp stream from which this sample was taken was regularly delivered into the spitzkasten. The volume of this stream

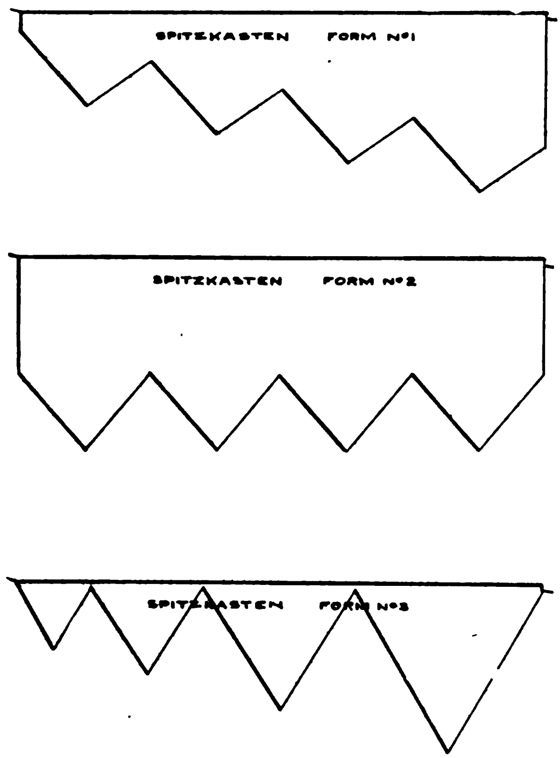


FIG. 2

was sufficient to fill the entire spitzkasten in about one hour, thus giving a current of 2 ft. per min. in a body having 35 sq. ft. of area. The water from the overflow was clearer than the supernatant water in the bottle, and still had but one hour for settling. A simple cause to which this may be attributed seems to be that in its slow motion the relation of water particles is continually changing, the more readily allowing the mineral particles to assert their superior gravity, while in a quiescent state the mineral particles become, in a way, lodged. Electric action has been suggested as a possible cause.

There are several forms of the spitzkasten, the various differences in form being due to difference in personal opinion. They all, however, have a general resemblance. Sketches of three forms are given. Fig. 2. These cover the ground in the order of their comparative theoretical effectiveness.

The first, form 1, is constructed with an increasing width and depth, allowing a gradually decreasing speed of current. As a result the settling tendency increases, and, as the cross-section of the flow is small at the beginning, the finer portions will be carried ahead the more readily to the compartment beyond, where it is designed they should settle. This form can be so designed that the cross-section progressively varies in such manner as to rapidly increase as the current passes beyond each of the divisions, giving a correspondingly decreasing speed, and as the pulp passes the apex or point of the hopper from which the draw-off is made, it has attained its minimum speed. From this point there is a practically uniform area of cross-section, to a point near the next division. It then decreases in area, slightly increasing the flow for a short distance, but the moment it passes this ridge the area increases rapidly as before to the point of the hopper, repeating this at every successive section with a decreased speed in each instance. The advantage of this action is obvious, and is clearly demonstrated in the figure showing this form of spitzkasten. (Fig. 2.)

The form 2 shown, is efficient and it was on a spitzkasten of this form that I made most of my observations as to action and effect. In this form it will be seen that the progressive sectional areas are increased and decreased in a series of identical fluctuations at each one of the sections, but there are two things which seem to assist in the classification and final settlement. One of these is that a portion of the content is drawn off at each successive spigot, and by just so much is the speed of the flow retarded. Another is that, in the matter of the finest of the material, the settling action is so slow as to hardly reach deep enough to be affected by the separating ridges until the pulp has traveled a considerable distance from the initial or feed end. Above the dividing ridges, is what might be called the effective sectional area of the moving body of water. Its sectional area in this case was 34⅓ ft. or about 86% of the total area figured to the apex. The spaces between the partitions can be said to contain practically dead water and consequently cannot be figured as effective, merely acting as a body in which the particles that are carried to it are at rest.

In the operation of this spitzkasten it was found that, when running normally, the water of the overflow end was, what might be called perfectly clear for a depth of about 14 in. from the surface. As the overflow was shallow and broad, the effect was that of drawing off from the surface without agitation. In this way the slime pulp was kept quiescent. In case any material is floating, and it is necessary to prevent this from going out with the overflow, a very shallow baffle-board could be placed across the

spitzkasten near the lower end, but it should not extend down into the body of water enough to cause any disturbance to the pulp.

The third, form 3, resembles the first in appearance only. It is composed of a series of independent inverted pyramids, the divisions between them being extended nearly to the top of the spitzkasten.

It would need no special argument or discussion to show the faulty action in this form, and reference need only be made to the face that what work may have been started in the first compartments in the way of settlement of the finer material is entirely defeated as the pulp is brought back to the surface and passes over each successive dividing partition. In this way the work has to be started all over again as each partition is passed, and effective settling is only done in last compartment. To the classifying action there would probably be less objection, but it can hardly be proved superior in this regard to the first form.

A new form of spitzkasten has been recently invented and patented by C. L. Buckingham, of Denver, which has some interesting features. The principal departure from the regular form is on the application of burlap screens in such a manner as completely to arrest the movement of the slime and to strain the overflow water. The entire top is covered with strainers and, as the head of the feed water is slightly higher than the plane of these, the water is gently forced up through. In addition to these strainers, others are placed underneath them in a sloping position, the action being to clarify the water to a great extent before it reaches the upper ones. It is a well-established fact that if any woolly, or coarse-woven fabric, such as burlap, is placed across the flow of water carrying slime, the minutest particles will attach themselves to the little filaments, and in time form a spongy porous mass which acts perfectly as a filter. This action would evidently take place on these screens, and experiment seems to prove that it does in this case. The arrest of the slime material is said to be complete, and on gaining sufficient weight it drops off, is carried ahead, and drawn off at the proper point. Draw-off cocks are placed at intervals and the pulp is taken from them at any desired solid consistence, as in the regular form. Another claim, which seems to be well founded, is the possibility of making a thorough classification.

When only settling and dewatering without classification is desired, simple conical or pyramidal tanks are often used. It is customary in such cases to feed at the centre of the tank, using the entire periphery for discharge of clear water, drawing off the thickened pulp at the apex. The Callow tank may be cited as an excellent example of this class. This tank is conical, having a height of 7 ft. 7 in., and an effective width at the top of 8 ft. I have found it possible to settle about 20 gal. per min. with this. The amount of solid material possible to handle in tank of this kind depends, consequently, on the consistence of the material fed into it. For instance, if the feed to the tank should be 10% solids,

from 20 to 25 tons per 24 hours could be readily handled and properly prepared. In cases where, from the use of hydraulic classifiers for instance, the consistence of the pulp should be as low as 4% solids, which has been met with in actual experience, the efficient capacity is materially reduced. In the use of the conical settling tanks the point giving the greatest amount of trouble is the tendency or liability to choke at the apex. This is due to the fact that everything of a solid character has to pass out through the opening at the point where everything is bound to settle. Avoidance of this trouble has been attempted in several ways, principally by the use of a system of pipes carrying a high water-pressure, by which the clogging material is forced back into the tank. This is at best only temporary, and when the pressure is removed the same condition is liable to recur.

A form of draw-off might be suggested for consideration that has been applied in some cases, though no statement of actual results is available. In this form there is an auxiliary chamber attached to the side of the tank that is in the form of a long tube having an opening into the tank for almost its entire length. The lower end is closed. At the upper is placed a gland through which is inserted a pipe, the lower end of which reaches nearly to the lower end of the auxiliary chamber. This pipe is used as the draw-off acting in the place of the ordinary goose-neck. In this manner the initial point of draw-off is removed a little distance from the apex. In case there is a clogging of the draw-off pipe, it is only necessary to withdraw the pipe sufficiently to reach a free flow, thus immediately re-starting the action of the tank. The pipe is then gradually returned to its original position, thereby cutting out the thick pulp that may have collected. Any foreign obstacle is more than likely to settle into the apex itself at some distance from the point of draw-off. The hose connected to the upper end of the pipe can rest on a roller or convenient support, so that in withdrawing the pipe the established height of the goose-neck will remain constant. A pressure pipe may be applied if considered necessary.

Final Treatment

It is with some delicacy that the subject of the final treatment of slime is approached, as there will appear in the discussion many facts which may be taken as an argument in the defense of the slime concentrating table which bears my name.

The line I shall take up will be, as nearly as possible, the abstract discussion of the principles involved in the construction and operation of various forms of concentrators used for this special purpose and the comparison of them from a practical as well as from a technical point of view. The purpose is to briefly review the previous statements and take up a few of the salient points, the more clearly to demonstrate the vital necessity of proper preparation of the pulp before concentration is attempted. The advantage of using the spitzkasten for the purpose of preparation has been emphasized already. An attempt was made to show the

various actions of settling, classification, partial concentration, and dewatering which are carried on simultaneously, as well as the special value of each in the proper preparation of the pulp for its final treatment. In comparison, instances were given in which, by the use of canvas tables, these advantageous results were not only in a measure lost, but that other conditions resulted that tended to endanger if not entirely prohibit good results in the final treatment. Another point emphasized was, that in any current whatsoever, losses are sustained. This radical statement was modified to the extent of an admission that in a very slow forward movement of a large body of water, a settling action is promoted that more than compensates for the slight loss which is inevitable.

Still another point dwelt on was that in the final treatment of slime, and it might also be added, in any process in which water is used as a conveying medium, the greatest possible consistency should be given to the pulp, compatible with complete stratification. In some characters of ore this consistency may be made greater than in others. For instance, clayey or porphyritic ores cannot be properly stratified at as high a consistency as can ores of a quartzose nature. While the former should not exceed 15 to 18% solid consistency, the latter may reach as high as 25 or even 30 in some cases. The point must be determined by experiment and no fixed rule can be laid down. In the discussion of the various forms of machines used for the purpose of slime concentration, one radical point has been assumed. This is that as a slime particle has very little or no ponderosity, any table or machine that depends on propelling these very minute particles across its surface by the aid of their inertia through using a bump, a reciprocating motion, or any similar method, cannot, from the nature of things, be classed as a slime table. The surface itself should have a continuous progressive movement conveying the pulp, which rests motionless upon it, through the various stages of stratification, washing, separation, and final cleaning of the surface. This is in direct compliance with the principle that in handling slime all disturbances must be most strictly avoided. It might seem inconsistent to state at this point that a vanning motion of some kind can be applied to the surface with a decided advantage to the results, both as to capacity and to efficiency. This seeming contradiction, however, can be readily harmonized by the statement that as soon as the slime particles are deprived of their liberty of action and are hemmed in and jostled by their neighbors they have less of an inclination to escape. In other words, when the pulp is dewatered to a high percentage of consistency, this very consistency, in direct proportion to its degree, allows of agitation without loss. It is because of this fact that in final treatment, the greatest possible consistency compatible to perfect stratification should be given to the pulp. However, even with high consistencies, a too violent disturbance should be avoided. Just enough motion should be given to keep the pulp 'alive' and give it an opportunity to stratify but not enough to mix it. This is a point

which is so often lost sight of by some who think that "if a little is good more is better." The character and effect of different forms of motion will be taken up under the proper head.

There are three radical types of tables used for the purpose of concentrating slime, classified as follows: (1) the conical buddle, (2) the end-moving belt, (3) the side-moving belt. In the first can be placed the Evans, the Roberts, the Cornish, and the Sperry buddles; in the second the Frue, the Embry, the Golden Gate, and the Johnston vanners; in the third the Luhrig, the Monnell, the McCoy, and the Roberts slime belts. There are a number of modifications and combinations of these types, for instance, in the A. & E. slime table which, having features common to two or more of them, can hardly be classed as one of radical type. In the discussion and comparison of the several forms of machines the subjects will be divided into: (1) the general forms, (2) the character of the different pulp distributions, (3) the efficiency of the different forms of motion and degree of consistency of the pulp, (4) the character of the separation on the different forms of machines, (5) the character of the products, (6) capacity, (7) efficiencies, (8) general discussion. In these comparisons typical machines will be selected: (1) the buddle, to which a circular vanning motion has been recently applied; (2) the Frue vanner, which is too well known to need any description; (3) the Luhrig belt machine, which was one of the original machines to adopt the side-moving belt. In referring to them only simple mention will be made.

(1) **General Form.**—The buddle is a low conical table, circular in form, with a surface which slopes uniformly from the centre to all points of its edge. It revolves slowly on its centre and has a quick circular vanning motion in an opposite direction to its revolution. The vanner is an endless belt, having a surface oblong in form, with a uniform slope in line with the greater dimension. It has a slow uniform progressive motion in line with its greater dimension, from the lower to the higher end and a quick uniform reciprocating motion transverse to the slow progressive motion. The Luhrig is an endless belt having a surface oblong in form, with a uniform slope transverse to its greater dimension. It has a slow intermittent progressive motion in line with its greater dimension which is supplemented with a slight knock or bump to assist in action of stratification.

(2) **Character of Pulp Distribution.**—On the buddle the pulp is delivered at the centre and assumes a fan shape which, in broadening, naturally decreases the depth of the flow, thereby retarding its speed. On the vanner the pulp is applied across its entire width near the higher and flows uniformly to the lower end. The sectional area being uniform at all points of the flow, its speed is practically uniform, with possibly a slight acceleration near the lower end. On the Luhrig the pulp is applied at the upper edge of the belt near the end from which it moves and flows across it,

assuming slightly a fan shape. In comparison, the fan-shaped flow and the consequent retarding of the speed, is obviously of advantage in the arresting of the slime particles, giving them more chance to settle to the surface of the belt.

(3) **Efficiency of the Different Forms of Motion and Degrees of Consistency.**—A number of tests have been conducted to ascertain the relative efficiency of the circular as compared with the simple reciprocating motion, as regards the degree of pulp consistency possible for the most satisfactory results. In every case it was found that the circular motion was capable of stratifying pulp of nearly twice the consistency of that which could be properly treated by the simple reciprocating motion. To be more exact, while the pulp fed to the vanner contained, on an average, 15% solids, that fed to the buddle carried 25%. In each case the consistency was about at the normal. With the Luhrig a much higher consistency can be maintained than on the vanner. The buddle and Luhrig more closely resemble each other in this regard. To explain: the pulp as it is fed on the vanner, across its width near the upper end, flows directly to the tail of the machine. This feed being continuous, the pulp must be sufficiently dilute to flow freely and not cake. During the flow from one end to the other the reciprocating motion of the machine is depended upon to stratify the pulp and bring the mineral particles in contact with the belt surface. These, by the travel of the belt, are carried back, up the slope of the surface directly in the face of the flow and passing over the upper or head end of the machine, are washed off into their proper receptacle. In the case of the buddle as well as of the Luhrig, the pulp is fed on the surface at the highest point. As it flows out on the surface it is spread out, and, in the case of the buddle, is allowed to rest. In the Luhrig it has not the same degree of opportunity to stop in its course, but certainly is not compelled to battle against a current moving in a direction opposite to the motion of the surface. After the pulp has been thus distributed, on the buddle as well as on the Luhrig, it is conveyed from the zone of feed flow and by the motion of the surface itself, is passed through the various stages of operation as before stated. As the flow of pulp on these two machines can be retarded to advantage, it is self evident that the degree of consistency is only limited by the efficiency of the motion employed for the purpose of stratification.

(4) **Character of the Separation in the Different Forms.**—Owing to the form of the vanner, the separation of the mineral from the gangue is largely regulated by the application of the washing water, assisted by the use of some of the mechanical adjustments. As there are two separate points of discharge, the head and tail, the products are therefore either concentrate or tailing, without any possibility of intermediate grades. With both the buddle and the Luhrig all discharges are made over one continuous edge. The tailing near the lower edge is first discarded, after which the

mineral or concentrate is washed off. In some cases the distinction between the two products is well marked, but ordinarily there is some merging. As a result of these conditions the possibility of cutting out various products is denied the vanner, while on the buddle and the Luhrig such divisions are readily made. While it must be admitted that the tendency toward merging prevents clean separation, especially when two minerals such as lead and zinc sulphides are being treated, still the possibility of cutting out as middling that portion inclued in the zone of merging, for the purpose of re-treatment, more than compensates. In this manner clean products can be obtained. In one case where a separation between pyrite and silica was made, a clean concentrate was cut out on the buddle. A middling product was also cut out between the concentrate and the tailing which was returned to the circuit and handled with the original pulp on the same machine. The product obtained ran 34% iron and 6½ silica, as against 29 iron and 18 silica in the product obtained from the other tables. These results total each in iron, sulphur, and silica 88% approximately, which indicates that there was some other element present, presumably zinc, in the form of sulphide. This advantage cannot be lightly passed over.

(5) **Character of the Products.**—This particular feature has been given considerable study and in its analysis, the particular claim of any machine, as to its being especially adapted to the concentration of slime, can be readily established or disproved. In all ores treated by concentration the mineral particles between 100 and 200 mesh will almost invariably vary in value from those which will pass a 200 mesh. In some cases it has been found that the finer size was higher in grade; occasionally the reverse is true. In one case in a slime carrying copper, the fine mineral particles were much lower in value than the coarse. In this case a buddle was being operated in a competitive test beside the vanner. By careful examination the following conditions were found to exist: the product from the buddle contained a large proportion of slime mineral; the silica content was practically the same in both products; a large quantity by weight was produced by the buddle; the product of the vanner ran 23% copper as against 18 in the product from the buddle; the total copper was greater from the buddle; and finally the tailing of the buddle ran noticeably lower than that from the vanner. Analyzing the conditions, it is clear that the presence of a larger proportion of slimed mineral in the buddle product, which at the same time ran lower in copper and practically the same in silica, indicated that the finest of the mineral was lower in grade than the coarser; the fact that the tailing from the buddle ran less in copper than that from the vanner indicated that the buddle made a closer saving, and this, coupled with the facts that the quantity of concentrate was larger from the buddle and contained a larger proportion of the slimed mineral, proved conclusively that the buddle was a better slime saver than the vanner. There are several lines of deduction which

might be reasoned out, but these are sufficient to make clear the first statement, that a study of the character of the products will readily prove or disprove the claim made for any machine as to its qualities as a slime concentrator.

(6) **Capacities.**—In the matter of capacity the types can be named in the following order, based on each foot of width: (1) the buddle, (2) the Luhrig, (3) the vanner. From experience with the three types the following figures of capacity have been deduced. They may be safely used for approximations: for the buddle, with a circular vanning motion, from 2 to 3 tons for each foot of radius; for the Luhrig, 1½ to 2 tons for each foot in width, and for the vanner, 1 to 1½ tons for each foot in width. These are all based on

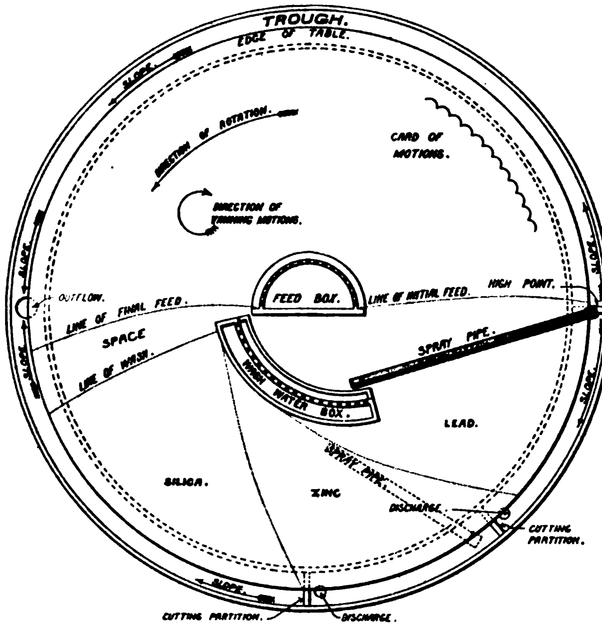


FIG 3. SPERRY SLIME TABLE

24 hours' running. This would give a 10-ft. buddle from 10 to 15 tons, a 3-ft. Luhrig from 4½ to 6 tons, and a 6-ft. vanner from 6 to 9 tons in 24 hours. The original Luhrig tables had belts 3 ft. wide, and it is fair to suppose that an increased capacity could be obtained by using a wider belt. The Wilfley slime table, which is a combination of the Luhrig and the vanner, has a much greater width, though the capacity is not increased proportionately. In analysis of conditions we can find a cause for these carrying capacities. It will be noted that they are about in direct proportion to the normal consistencies of the pulp, as discussed under that head.

(7) **Efficiencies.**—A few statements of results of actual work as practical illustrations of some of the previous statements are

here given. In one case, where the three typical machines were operating, the results were as follows. The buddle had a capacity of 10 tons, consistency 25% solids, concentrate 34%, 6½ silica, tailing trace in gold. The vanner had a capacity of 6 tons, consistency 15% solids, concentrate 33% iron, 10 silica, tailing 0.02 oz. gold. The Luhrig type, 6 tons capacity, concentrate 29% iron, 18 silica, tailing 0.01 oz. gold. In another case the buddle had 12 tons capacity, 25% solids, concentrate 47% lead, extraction 67%. The vanner 6 tons capacity, 14% solids, lead 55%, with 39% extraction.

(8) **General Discussion.**—The question may arise as to what might constitute a proper amount of motion. As the result of a number of experiments it is thoroughly determined that in properly stratifying slime pulp, 300 inches of motion per minute is the maximum and that should only be used, either when heavy or clayey gangue is to be separated from a heavy mineral, or when two minerals are to be separated from one another. It is ordinarily the case that over 200 inches per minute are used. In the vanner, for instance, 200 revolutions of the shaft is an average speed. With this speed the throw is from ½ to ⅝ in., making from 1 to 1¼ in. per complete revolution, giving from 200 to 225 inches per minute. In the buddle with ¼-in. circle of motion, 300 r.p.m. would give 235 in. per minute. With ⅜ in. circle at 250 r.p.m. would give 281 in. per min. These sizes and speeds have been shown to include those giving the most favorable results, and so it can be stated that between 200 and 300 in. per minute will be found a speed which, in almost every instance, will meet the requirements.

In discussing these three types, only such forms have been taken up as might be considered as most commonly used, and what might be called standard or representative. There is another form which might be considered as a fundamental type. It is that using the principle of the pan. This has never proved satisfactory or practicable, as it has never been made automatic and continuous. It was employed in the Copeland table some years ago, but was not successful. The A. & E. slime table, recently introduced, seems to be the most practical application of this principle. This is a combination of the pan and the Luhrig. It certainly presents interesting features. The Wilfley slime-table is constructed on the general form of the Luhrig. So much so, in fact, that it has been considered as a form of this type. It has some features, however, which put it at variance with the Luhrig. The up-throw of the motion, the use of transverse trays, and the method of delivery of the concentrate give it some resemblance to the vanner. With the motion, the tendency is to throw the mineral to the upper ends of the trays against the ascending current.

(December 10, 1910)

The Editor:

Sir—In the first part of E. A. Sperry's paper on the above subject (*Mining and Scictific Press*, August 6, 1910), he devotes considerable space to the different machines in use for pulverizing ore,

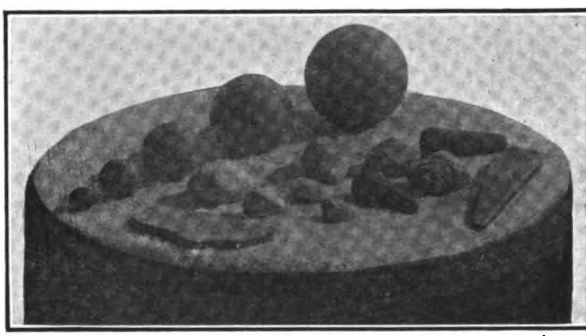
and their action, and makes a few remarks on the products. As he says, rolls exemplify pressure action, and stamps impact, while pressure-impact is utilized in Huntington and Griffin mills. A number of the former machines are in use here, crushing the soft oxidized ores. They deal with up to 40 tons each daily, very effectively. As to the Griffin, the South Kalgurli, Perseverance, and Great Boulder mills had 4, 16, and 13 of these, respectively, and from the two former mills they have been discarded in favor of ball-mills, while additions to the latter mill are made by installing ball-mills. The Griffin works by pressure-impact, as ore in the mill is found to have a sharp fracture. Dynamite, which finds its way into the mill, explodes violently, generally breaking the pan and suggesting impact. If the mill runs empty, the muller bangs hard on the die-ring, and often the spindle breaks. The Griffin product is fine and up to 70% will pass 150 mesh, being crushed through a 15-mesh screen. These mills are rather costly to run, and are not favored generally.

In describing pressure-torsion machines, Mr. Sperry does not mention the grinding-pan, which no doubt comes under this head. The shoes on the muller-plate of the pan press heavily on the dies, and as they are revolved, a dragging or twisting motion results, the ore particles being crushed by pressure and dragging or twisting set up. The pan is a good grinding machine, and with feed of which only 45% will pass 150 mesh, 85 will pass the same screen after grinding. About 75 five-foot and 19 eight-foot pans are in use on this field, most of them grinding roasted ore, the balance working in series with the tube-mills on raw ore.

The tube-mill is probably the best example of impact-grinding. I understand the action of this mill to be one in which the pebbles are carried almost to the top of the mill, and fall in a shower to the bottom, striking other pebbles at the bottom, particles of ore being crushed by the impact. A tremendous amount has been written on the tube-mill, but I have never seen details of a microscopical examination of the slime particles, whether they show a sharp fracture or otherwise. Of course, a great number of pebbles are simply carried up a certain distance, and then slide back again, or roll over one another to the bottom. Pebbles doing this, would grind the ore more or less. In 1904, the Krupps in Germany made a number of experiments on the action of the tube-mill, these being described by H. Fischer in several technical journals soon afterward. Photographs were taken of the pebbles in action in a mill made for the purpose, and these showed the impact action quite clearly. It would seem that the plain type of liners for the tube are not as good as the corrugated or rough type mostly used now, for the reason that the latter toss the pebbles about more, giving more chance for the impact of one pebble against another, while the smooth liner does not tend to carry them up far, irrespective of centrifugal force.

Where I do not agree with Mr. Sperry, is in regard to the ball-mill. He said: "The ball-mill is a modification of the tube-

mill with the substitution of steel balls for pebbles, and steel lining in place of the silex brick. It is designed for dry grinding, and is used extensively in cement manufacture." It is doubtful whether the Krupp ball-mill, which seems to be the most efficient of all the ball-mills, is used anywhere with greater efficiency than it is in Kalgoorlie. There are here at work 27 No. 5 size, which crush up to 45 tons each per day, and 17 No. 8 size, which crush up to 90 tons each. About 45% of the product from a 25-mesh screen on the mills will pass 150-mesh. Now, I fail to see how the ball-mill is a modification of the tube in any way. According to size, the Krupp mill is charged with 1 or 2 tons of steel balls, and revolves at 21 to 25—in one case 24—revolutions per minute. The ore, crushed by the balls, passes through the perforated grinding-plates, and is screened by the coarse or inside punched screens, the oversize being automatically returned to the interior of the mill, while what passes the inside screen is screened by the outside or fine-wire



BALLS, STONE, AND PIECES OF STEEL FROM BALL-MILL

Note Rounded Shapes and Especially the Perfect Roundness Maintained by the Balls While Wearing Down.

screens, the oversize also being returned to the mill. The working of the mill is ingenious, to say the least. The grinding action of the balls is not due to impact. The only impact that might take place is when the balls drop from one grinding plate to another, a distance of 4 inches in a new mill. The action of the balls is one of pure rolling, grinding, and abrasion, the balls being lifted up level with the mill shaft or axle, and then simply rolling back, tumbling over one another, rubbing and grinding away the ore between them. This is not mere supposition, but fact, as I have actually seen the action. At the Associated mill there are four No. 5 Krupp mills crushing oxidized ore, which contains 5% moisture. Very coarse screens are used on this, as the fine screens, 25-mesh, would be clogged. In crushing this ore no dust is made, so by the aid of a portable electric light, placed well down in the feed hopper, the balls can be seen working. Repeated observations show that the balls and ore are carried up about level with the mill shaft, and

having reached this point, start to roll back, and the whole 1200 lb. of balls (light weight being used on this soft ore) roll and tumble about, grinding the ore by abrasion. It is a most interesting sight, fully proving that there is no impact grinding. To further prove this, it is only necessary to take note of the ore in the mill when it is stopped for repairs. Down to the finest particle, it is round and smooth, just as if water-worn from a river, where stones are quite round from rolling. Pieces of drill steel from the mine are fed in at times, and these become rounded in the mill, although they hinder the good work of the balls to some extent. The Krupp mill works either wet or dry, but is especially efficient as a dry crusher. To support my theory, I enclose a picture showing the perfectly round form the balls keep in wearing from full size to the smallest, the rounded pieces of ore and steel, the latter originally hexagonal, and two pieces of plate which do not add to the grinding efficiency of the mill.

M. W. von BERNEWITZ.

Kalgoorlie, Western Australia, October 10.

CYANIDING SULPHIDES

(July 30, 1910)

The Editor:

Sir—There is a subject about which little or nothing has appeared in the technical press. I refer to 'Cyanidation of Sulphides,; and should be pleased to gain some data. It would be a boon to the mining world if those who have been engaged or are at present engaged in treating sulphides, you might say pyrite, by cyanidation would diffuse some knowledge on the subject by giving their experience. There is ample room, I think, for a few articles on the subject, describing fully the kind of sulphide treated, time of contact, strength of solution, aeration, and methods of precipitation. There are some plants in California and Nevada treating their sulphides at home, and information regarding their experience would be useful to those now undertaking the task, as sulphides treated and difficulties to overcome elsewhere may be similar. Instead of shipping to smelters, the treatment of the baser products at home is becoming more in vogue. In most cases it is cheaper, especially at a mine having a cyanide plant. Again, it solves the difficulties and expense of transportation. Any information that may be given from whatever clime will not only be added to metallurgical knowledge, but will be greatly appreciated.

INGENIERO KCN.

San Francisco, July 19.

(September 3, 1910)

The Editor:

Sir—This is not only a subject of interest, but of importance to the mining world. For the last seven years I have studied and experimented along these lines, meeting with encouraging success.

In some cases results were almost unbelievable, in fact disputed, but I produced the bullion at the 'clean-up' which corresponded with assays, and that is, after all, the most convincing argument. I do not claim to have discovered a new process, but the successful application of the bromo-cyanide process. At three different well known mines where I have installed plants, different treatments are in vogue. The same strength of solution and quantity of bromine does not apply to all sulphides, but, like treating different tailings, a specific treatment is of course essential.

I shall only cite in this letter the treatment I employed on the concentrate at the Black Oak mine, Tuolumne county, California, in 1905; this being the most interesting on account of the high content of gold and silver. The concentrate contained sulphide iron (pyrrhotite), galena, and copper. Occasionally the content of galena ran as high as 4%, but the copper never exceeded 2, and was generally nearer 1%. The silver was mostly in the form of a sulphide. There was about 3½% of sulphide in the ore, which, when concentrated, represented six different products, ranging in value of $100 gold, 20 oz. silver, to $2000 gold, 160 oz. silver per ton. Only about 500 lb. of the latter product was made each month. It was all cyanided and turned into a bar, averaging an extraction of 99% and over, on 30 tons (a month's run), at a cost of $5 to $8 per ton, the cost depending on the value of the tailing. The Frue vanner product (designated as No. 1), was the coarsest material treated and showed beyond all doubt the advisability of fine grinding; for, on comparing the tailing value with that of the tailing from other products treated, which were finer, the No. 1 tailing always ran $1 to $2 higher. While these were the coarsest treated they could be considered fine in comparison with the average of concentrate, for the mill crushed to 60-mesh. The average value of No. 1 concentrate was $100 gold, 20 oz. silver per ton.

The treatment began at the machine where concentrate was produced. If, at the vanners, the sulphides were hoed out into the front settling box, and let stand for eight or ten hours. Every half hour during that time they were pounded with a shovel. They gradually packed and settled under this treatment, so that at the end of eight hours they were very hard and solid and contained only 12 or 14% moisture. The water was then bailed off and sulphides shoveled out. The product is easier to handle, there is less loss, no extra cost, and this method is to be recommended on the ground of its neatness alone. On the accumulation of 3000 lb. or thereabout, the concentrate was trammed to the cyanide plant, spread out in the sun, and dried to about 2 or 3% moisture.

Winter conditions, of course, interfered with this treatment, necessitating the discarding of this preliminary operation which always resulted in a little higher tailing loss. Steam drying improved this a little, but nothing we could do would bring extraction up to that on a sun-dried charge. My theory for this is that the sun in drying out the moisture, split or fractured the particles, making them more porous, and the sulphides took up more or less

oxygen in drying, and being dry, helped the solution to penetrate by absorption. The charge treated by agitation, consisted of 3000 lb. net weight (carefully weighed, moisture determined, and sampled for 'heads' assay), 2000 lb. net weight of water (solution), about ½ lb. of liquid bromine, and 40 lb. of cyanide (2% solution).

The agitation tank is of special construction. It should be of steel in preference to wood, or wood, steel lined. In fact steel should be used instead of iron, wherever a wearing surface comes in contact with pulp. I advise the use of steel shoes and dies in all mills where cyanide is used. A higher extraction and less cyanide consumption, is my experience. The agitator used was made approximately air-tight, so as to have the charge, while agitating, under a slight air or gas pressure. It has a trap door in the top for the convenience of loading.

After sulphides and water are in the agitator the bromine was added and the trap door closed immediately to retain fumes which, while heavy, would soon be fanned out and lost. I believe these fumes perform just as much work as the liquid dissolved in the water. The charge was agitated a half hour before cyanide was added, and the air started, putting the charge under a slight pressure which was maintained during the whole of the agitation. About four hours after adding cyanide the largest consumption had taken place. The extraction at that time was about 90%, 18 hours more being required for the remainder. At the end of the 22 hours' agitation the charge was discharged into a rectangular vacuum tank, and then solution was drawn through the filter, leaving the sulphides, with about 20% of the pregnant solution. This was displaced by wash-water. On being thoroughly washed the tailing was sampled for assay, and then shoveled out. Seldom more than 30 of the 40 lb. of cyanide would be consumed; more often 20 lb., sometimes as low as 10 lb.—this depending altogether on the quantity of magnetic iron present, and the amount which went into solution. The gold was invariably extracted regardless of large or small cyanide consumption. The silver was always more or less of a worry. If the cyanide consumption was small at the end of the 22 hours, the silver was always extracted. If cyanide was nearly all destroyed, the solution would be dark, and the silver lost. This seemed to be due to the iron going into the solution and precipitating the silver. This could only be avoided by constant vigilance toward the end of the treatment as it was at that stage the trouble would occur if at all. The average tailing value on No. 1 sulphides when not sun-dried before treating, was $2 gold, 2 oz. silver per ton. These same sulphides sun-dried, averaged $1 gold, 1 oz. silver per ton.

The other products were treated in much the same manner with about the same strength cyanide and bromine. Although much richer they were easier to treat. The reason for this was their extreme fineness. No. 2 product, which was caught on the canvas plant, ran $250 gold, 50 oz. silver per ton, received the same treatment as No. 1, and gave an extraction of from 99 to

100%. Seldom more than 60c. was left in the tailing. When the material was sun-dried, often only a trace was left. Another canvas plant product (No. 3) ran $150 gold, 30 oz. silver per ton, and gave the same results as No. 2 with less cyanide, 1.5% solution and the same quantity of bromine being used. The high-grade concentrate, $2000 gold, 150 oz. silver per ton, was obtained from the side settling boxes on Frue vanners where the very fine sulphides would float and finally settle. These were either treated by themselves or mixed in with the No. 2 product. Zinc shavings was the means of precipitation. The solution, while always strong in cyanide, was built up in alkaline strength with caustic soda, 2 lb. to the ton, before entering zinc boxes. A small steady stream should flow in the boxes. Sudden starting and stopping will cause a loss by loosening the adhering particles and carrying them on down. About 95% of the gold and silver will precipitate in the first three boxes, but from 10 to 18 boxes are used as a precaution against loss. I have never yet allowed more than a trace to be found in 10 assay tons of solution on leaving the zinc boxes. I do not believe that a higher tailing solution value is necessary in any plant. The zinc-box tailing solution may be used again on making up a new charge, bringing its strength up with additional cyanide.

These were the results at one mine, but I have had the same success at others. In one case the concentrate was three or four years old, badly decomposed, and assayed $30 to $40 gold per ton. There was 700 tons piled up—it costing as much as the material was worth to ship to smelter. The mine was 60 miles from the railroad. This concentrate was treated at a cost of $10 per ton, including the tailing loss. There are many other mines in similar situations where concentrate is not worth saving owing to the high cost of shipping. The cost is not all in freight and smelter treatment charge. Sacking, hauling, and the wear on sacks, is quite an item. Taking $100 concentrate as an example—95% of the assay value is paid, $5 being lost to the shipper; $20 per oz. is paid; were it bullion the price would be $20.67; and on 5 oz. the loss is $3.36 more. It is not necessary to review the moisture disputes and dissatisfaction, but summing up the total cost, it will be found that a liberal margin can be allowed for the treatment of concentrate by cyanidation.

MURRAY N. COLMAN.

San Francisco, August 10.

(September 24, 1910)

The Editor:

Sir—One of your recent correspondents asks for information on cyaniding concentrate. Following is a list of articles in which the subject is mentioned, which may interest him: *Bulletin* Amer. Inst. Min. Eng., S. F. Shaw; *Mining World,* June 1, 1907, Frank C. Smith; *Eng. & Min. Jour.*, April 3, 1909, Walter Brodie; *Journal* Chamber of Mines Western Australia, May, 1908, B. L. Gardiner; *Mining and Scientific Press,* October 3, 1908, A. E. Drucker; *Min-*

ing and Scientific Press September 26, 1908, F. C. Brown; *Eng. & Min. Jour.*, April 17, 1909, E. Walsh; *Mining and Scientific Press*, October 2, 1909, J. D. Hubbard; *Mining World*, November 27, 1909, Etienne A. Ritter; *Mining and Scientific Press*, March 19, A. E. Drucker; *Pacific Miner*, A. C. McIntire. Beside the above there are a great many short notes and quotations that give valuable information. If I were sure that I have correctly guessed the identity of your correspondent, I would dare suggest that there are just ten mining journals which should be considered indispensable on a big milling and cyaniding plant. One manager subscribes for nineteen such publications and says it is the best investment he has made. He has a list of sums saved which he credits to what the shiftmen read in the little library, and if something substantial is not added every week he starts an investigation.

MARK R. LAMB.

Milwaukee, Wisconsin, August 29.

CYANIDATION AND SMELTING

(Editorial, August 27, 1910)

Cyanidation of concentrate involves many difficulties which are being overcome in many ingenious ways. In March, Mr. A. E. Drucker described for our readers the methods used at Taracol in Korea, and in this issue Mr. F. C. Brown, recently manager for the Waihi Grand Junction Gold Company, Ltd., contributes from experience in New Zealand. Interest in the matter is world-wide and the practice is becoming increasingly common. It rests on solid economic ground. It is not only that by shipping bullion in place of concentrate the miner is independent of the smelter, and is paid for his gold at the rate of $20.67 per ounce in place of $19.50, but despite resulting waste of base metal, there is usually an actual economic gain. Smelters must base their charges not on the cost of handling a particular ton of ore, but upon the cost per ton for furnace mixture. Accordingly smelting rates are adjusted so as to best bring out steady shipments of silicious, lead, iron, and copper ores, in the proper proportion for economical handling in the furnace. As the processes of treatment and marketing require time, and steady running necessitates carrying large stocks of ore, interest charges on ore purchased naturally must be taken into account. It follows, without any necessary implication of unfairness on the part of smelter managers, that the rate on any particular ore may be high or low as compared with the actual cost of reducing it, and further that smelting rates on any given sort of ore only meet corresponding rates for treatment by other processes when the smelter needs that ore to preserve the proper ratio in the furnace. Since he must in any event melt much barren material, the cost of doing so has to be assessed against the ore that can best be spared. The basis of fixing an individual rate is therefore the same as in railway traffic—'the value of the service' or 'what the traffic

will bear,' depending on the point of view. It is clearly to general advantage that the amount of barren material sent to the smelters should be as small as possible. If therefore a concentrate does not contain sufficient lead or copper to pay the cost of reduction, it may or may not put a burden on other ores, according as its excess of silica or iron is desirable or the reverse. Sulphur is negligible since under present conditions the amount in lead and copper ores alone is greatly in excess of commercial requirements. Not only may a concentrate such as noted impose a burden upon other ores smelted with it, but its handling and transportation involves unprofitable labor. The freight bill of every civilized country is enormous and it is good economics to ship in the most condensed form and that nearest the finished product, whenever possible.

These natural conditions have been complicated by others that are artificial and in any discussion of smelting rates the ratio of capitalization of the smelting companies to tonnage handled must be taken into account. It is frequently assumed that capitalization is properly measured by the cost of replacing useful plants. This is, however, by no means the whole measure of value. Large stocks and deferred sales require abundant liquid capital and the money spent in fruitless experiment is a proper charge on operating expense if the industry is to progress. These and other items properly enter into consideration of charges. Making all proper allowance, however, there seems little escape from the conclusion that the smelting companies have been over-capitalized and that in general too large a part of the profits of the industry has gone to the smelters rather than the miners. Mining involves more hazards than does smelting, and miners should therefore have the larger portion of the speculative profits. Since the security for return in smelting is greater, the rate of profit should be smaller. In America at the present time, this matter is complicated by the fact that practically all of the smelting companies have interests, direct or indirect, in mines, and it is not altogether easy to properly apportion the profits. In Colorado there was for many years an excess of silicious ores. The rate accordingly was made high on ores of that character and low on others. Naturally the first concern of the big smelting companies as they were organized was to assure a steady supply of the then less abundantly mined fluxing ores; which was done by purchase of mines. It was but human that the rates should be continued low on the fluxing ore and as much as possible of the cost of smelting assessed on dry ores. It was equally natural that production of lead and copper ores increased and that miners of silicious ore were driven to develop first chlorination and later cyanidation until the smelting rate on dry ore came to have no relation to the cost of treating the same ore by hydro-metallurgic processes. The obvious course would have been to change smelting rates so as to throw more of the cost on the fluxing ore; thus encouraging production of dry ore and discouraging production of the others. An alternative would have been to reduce rates as a whole. To do either, in any large way, would, however, have inter-

fered with the profits of the enterprise as a whole, and naturally the managers preferred to try for other ways of meeting the situation. As a result the furnaces have been run at half capacity for some time in the hope of the discovery of additional silicious ores containing sufficient gold and silver to permit payment of a smelting charge. This was the natural and the conservative course, and there are good reasons for it aside from any selfish interest in immediate returns. Any arrangement that would permit unlimited production at Leadville and in other districts from which the fluxing ores come, would influence prices of the base metals, perhaps disastrously. Gold, however, is not subject to fluctuation in price and its output can be safely increased. Silver also remains valuable even though it is a commodity. Continued production of the gold and silver is profitable even though the world is not ready to absorb an indfinite amount of lead, zinc, and copper. The hydro-metallurgical processes have therefore flourished, and now, having been shown to be widely adapted to treatment of dry ores, are being applied to getting the gold and silver out of the complex sulphides that form the bulk of the concentrates. The fact that cyanidation, in particular, may be economically carried on in small plants is to its advantage, but fundamentally we believe that the present ascendency of the hydro-metallurgical methods of treatment of gold and silver ores on the greater demand for gold and silver than for the base metals. A reduction in rates sufficient to allow the smelters to compete for the dry ore, even if possible, would result in an increased production of the base metals that would be disturbing if not disastrous.

CYANIDATION OF CONCENTRATE

By F. C. Brown

(August 27, 1910)

Probably most ores will come under the class of ores containing gold and silver finely disseminated through the various mineral constituents, as there are but few cases where all the valuable metal is in the form of free gold and silver, or where the gold and silver is in such forms as to be easily dissolved by cyanide solution. The ores and cases to which I especially wish to direct attention are those where the metals, other than gold and silver, if recovered in the form of concentrate, are not of sufficient value to cover shipping and treatment rates on the concentrate. In these cases, and there are no doubt many of them, it is advisable, if possible, to treat the concentrate at the mine for the recovery of the gold and silver, and allow the lead, zinc, copper, and other metals, to go into the tailing.

For the last three years I have been at a mine in New Zealand where just such conditions were present. The ore consists of a gangue of quartz and calcite containing the following minerals finely disseminated through it: iron pyrite, zincblende (chiefly the black variety), galena, some copper pyrite, and traces of arsenic, antimony, and selenium. By analysis it was estimated that 8 to

10% of the total weight of the ore consisted of these minerals. The gold and silver value of the ore was about $9 per ton (of 2000 lb.), about $1.50 of this being silver. There are no smelting concerns in New Zealand. Any concentrate produced has to be sent at heavy expense to Australia, and it was found that although 40% of the total value of the gold and silver content of the ore could be recovered in the form of concentrate there was little profit from the sale of the concentrate after paying bagging, carting, shipping, and treatment charges.

Upon taking over the management of the mine I at once made arrangements for treating the concentrate at the mine, and a special cyanide plant was erected for this purpose. The results were highly satisfactory as it was found that a 90% recovery was effected at a fairly reasonable cost.

The concentrate treatment consisted of the following steps: (1) regrinding in tube-mill; (2) agitation by compressed air in Brown (Pachuca) tanks with fairly dilute cyanide solution; (3) vacuum filtration. Great difficulty was experienced in grinding the concentrate sufficiently fine. Some idea of the fineness necessary may be obtained from the fact that, before regrinding, all the concentrate would pass 200-mesh sieve and it was estimated by time settlement-tests made in water, that the reground material would all pass 400-mesh if such a sieve were obtainable.

A fairly large tube-mill was required for about 10 tons of concentrate per day (24 hr.), so it can readily be seen that the power consumption per ton was high, as was also the wear of pebbles and liners. After running the plant some months I came to the conclusion that such fine material by itself is too fine for economical grinding in tube-mills, and this opinion was strengthened by the fact that later, although very much finer grinding of the ore was tried and consequently finer concentrate produced, the percentage of extraction after regrinding was lower than when the ore was being crushed coarser. From this I inferred that the concentrate was not being reground as fine as before.

In order to overcome this difficulty it was decided to try adding coarse sand to the concentrate as it entered the tube-mill, the idea being that this gritty material would help the grinding, in the same manner that the barn-yard fowl picks up grit to grind its food. The result of this experiment was even beyond our anticipations, as it was found that the extraction was increased some 3 or 4% and the wear on liners and pebbles was materially reduced. It next occurred to me that the concentrate might advantageously be reground in the tube-mills used for grinding the ore, provided some simple method could be devised for continuously feeding it to these mills, there being always a good supply of coarse gritty material in these mills to assist the grinding. After numerous experiments such a method was devised and the success of the scheme was all that could be desired. This meant the abandonment of the special concentrate-treatment plant. The total extraction on the ore was higher than when the concentrate was treated separately. It is

now about 90%, and the costs for labor and chemicals are considerably lower. This method of treating the ore and concentrate together has already been brifly described (the result of a communication from me), in an article by A. Grothe, president of the Mexican Institute of Mining & Metallurgy, which was published in the proceeding of the Institute for August, 1909, 'Cianuración de Concentrados,' but this further reference to the method will be of interest to some. I have come to the conclusion since arriving in this country, that there are many mines in the Rocky Mountain regions situated away from the railroads, which, if they had a plant at the mine that would give a high extraction of the gold and silver content of the ore in the portable form of bullion bars, would give the owners materially increased profits compared with the present system of shipping the concentrate. This method of treatment might also bring about the resumption of work on some mines and the opening up of others.

(October 1, 1910)

The Editor:

Sir—The article in the August 27 issue 'Cyanidation of Concentrate,' by F. C. Brown, was particularly interesting and instructive, coming from an engineer whose reputation is world-wide, and who has done so much to bring the cyanide process to its present state, nearing perfection, both chemically and mechanically. Mr. Brown's article deals with finer grinding of the concentrate. I really wanted information about the process and results obtained when the concentrate was not reground. I will admit it is a crude way to treat it, but I plead 'not guilty' to the arrangements. A great difference of opinion existed at a certain plant, as to the best way to feed the solution, also the quantity. I shall explain the plant to be more clearly understood.

The ore was crushed through 20-mesh so called, amalgamation inside and on 12-ft. outside plates. Concentration was effected on a table and a 6-ft. vanner. There was no classification. The tailing went to waste, the concentrate to the tanks, by sluicing. The ordinary different length arm distributer was used, but it did not revolve, as the wheels of the carriage had been removed to allow of more grade from the concentrators to the tanks. This was revolved at times to allow a more even distribution in the tanks. A tank was filled in about 10 or 12 days. Lime was fed to neutralize acidity, but the amount I cannot give, on account of conflicting opinions; the alkalinity was kept at two and one-half, and the standard solution was 8 lb. per ton. The latter was used continuously and re-standardized. An argument arose as to the way of feeding the solution. The 'pseudo' in charge claimed that a better extraction would be gained by feeding a certain amount. I must state that solution was fed in the tanks for 12 hours and the other 12 was reserved to drain same. No account was kept of the amount of solution pumped, or used. The millmen pumped the solution on night shift when the storage tank was empty. The other opinion

was that a better extraction would be obtained by using all the solution that could be run on for the reason that the chief aim was the greatest possible contact between the gold content of the concentrate and the percolating solution. Will any engineer who reads this kindly give his opinion?

Sands, of course, would present a different proposition, but in the treatment of this concentrate I personally am inclined to agree with the latter statement given by the man running the plant, and not with the 'pseudo' in charge, for this and many other reasons. The young man in charge was decidely better qualified to express an opinion, even if he should be wrong in this case.

Again, if too much lime were fed on the tanks, would it tend to form a slimy, or colloidal matter on the zinc shavings in the boxes? It was found that this only occurred when the gold tanks became empty, and the colloidal matter in the bottom was allowed to run into the zinc boxes. The gold tanks were absurdly small, and often the flow into the zinc boxes had to be cut down to prevent this. In both cases the amount of precipitate obtained was cut down. Is this matter in the bottom of the tanks caused by absorption? And would the solution so often used carry impurities which would cause a loss in the extraction by imperfect contact in the tanks and precipitation in the zinc boxes?

INGENIERO KCN.

San Francisco, September 30.

The Editor:

Sir—From all that I can learn concerning the treatment of concentrate by cyanidation, the greatest success attained has been after fine grinding, taking care always to expose the concentrate as little as possible to oxidizing influences. The experiments of Dr. Schidell at the Utica mine at Angels, California, those of W. G. Scott at the Black Oak mine, Soulsbyville, California, and of Mr. Diggles and others at the Melones mine, Robinson, California, all resulted in securing the highest extraction from finely-ground sulphides. In some instances as high as 96% extraction was reported as obtained from concentrate from canvas tables whereas the highest extraction on vanner concentrate seldom exceeded 72% of the gold and even less of the silver present. Experiments reported by others indicate that whatever may be obtained from ordinary concentrate, such as is generally recovered from concentrating machines, a much higher extraction is possible on the same material when it is finely ground. On low-grade sulphide it may not pay to regrind prior to cyanidation even if a higher extraction is obtainable by that means, consequently it resolves itself into a commercial problem, as well as a metallurgical one, for there are few metallurgists who take pride in carrying their operations to a point where the cost of recovery of metals exceeds their commercial value. Often a process is worked out successfully, when viewed from the chemical or metallurgical standpoint, but which is too expensive to admit of its application along commercial lines. The

metallurgist then continues his experiments with a view to reducing the cost of the operation, and not infrequently he, or, as likely, some one else, succeeds in this and the 'impossible' process becomes a recognized commercial possibility and gladly accepted as such by all who have use for it.

Seattle, September 22. CYANICIDE.

(December 24, 1910)

The Editor:

Sir—In your October 1 issue, 'Ingeniero KCN' asks for information about the treatment of unground concentrate. Sulphides direct from the vanners are being successfully treated here by percolation. Little headwater is used on the machines and about 50% coarse sand is allowed to come over with the concentrate. The material is dug out of the boxes and trammed directly to the vats. Lime is mixed with the charge, the amount varying from 6 to 9 lb. per ton of concentrate. Nothing is gained by giving water washes after addition of lime, as all ferrous salts dissolved will be re-precipitated as hydrates. The ore here contains a considerable amount of arsenopyrite, and a highly alkalinity is maintained. About 0.50% titrating with deci-normal oxalic acid gives the best results. Pumping solution on the charge intermittently and allowing it to drain probably helps the extraction, but the method is slow. Blowing air up through the charge is more efficacious. Transferring from one vat to another is to be recommended where cheap labor is available. Excess of lime tends to form an incrustation in the pipes. The first solutions coming off will sometimes carry lime compounds in suspension, which, if allowed to run to the precipitation boxes soon coat the zinc and prevent contact. I find that the only way to deal with these turbid solutions is to run them direct to the sumps, passing the zinc boxes, and pumping up again when adding strong cyanide to the stock tank. In the treatment of concentrate here the first solutions coming from a vat contain no free cyanide and carry little gold, hence they are run to waste. When necessary, water is added to the stock tanks to supply the deficiency. By this means the solutions are kept in good condition. The concentrate in question assays from $9 to $10 per ton and a 65% extraction is obtained. The ore is crushed through 15-mesh punched iron screens.

G. CHESTERFIELD EVANS.

Kuk San Dong, Korea, November 17.

(January 21, 1911)

The Editor:

Sir—In view of the articles and discussions on 'Cyanidation of Concentrate' appearing in recent numbers of the *Mining and Scientific Press,* the following particulars of practice at the Waihi mine,

New Zealand, may prove of interest. The concentrate (from Union vanners and Wilfley tables) consists chiefly of iron sulphide with a small proportion of zinc, copper, and lead sulphides; it is shoveled from the collecting-boxes into tubs holding about 600 lb. wet weight. Care is taken to keep the concentrate under water and the product from each of the three mills is daily sent to the concentrate-treatment plant. Some 500 tons are produced each month assaying about 5.5 oz. gold and 65 oz. silver per ton. The concentrate is ground in tube-mills and delivered to spitzkasten, the coarse returning to the tube-mills and the fine passing to dewatering boxes. It is difficult to say just how fine the concentrate is crushed, but when ready for treatment it is in an impalpable condition. Agitation by air is carried out in conical-bottom tall tanks, the solution being kept at an average strength of 0.4% KCy. Time of agitation varies from 8 to 10 days. The solution is then separated by filter-pressing, and bullion precipitated by the zinc filament. No trouble is ever experienced in obtaining satisfactory precipitation. Consumption of sodium cyanide averages 16 lb. per ton; this represents a consumption of 0.25 lb. per ton of original ore. The cost, including labor, repairs, and renewals, power, transport, cyanide, zinc, flints, and sundries, is about $6.25 per ton; this represents 10c. per ton of original ore. The extraction has been uniformly good, as shown by the following figures:

Year.	Tons.	Gold, per ton. Oz.	Silver, per ton. Oz.	—Extraction— Gold. %	Silver. %
1904	1,992	7.65	98.25	95.1	92.0
1905	3,719	6.65	96.35	95.5	93.3
1906	4,692	5.68	81.00	95.6	94.7
1907	5,581	5.40	66.75	95.7	94.3
1908	6,061	5.44	66.00	96.3	93.0
1909	6,339	5.60	63.30	96.3	93.5

E. G. BANKS.

Waihi, New Zealand, November 22.

CYANIDATION WITHOUT CONCENTRATION

(Editorial, February 11, 1911)

Cyanidation without preceding concentration of the slime is being carried on at the mill of the Tonopah Extension Company, the tube-mill product going direct to the cyanide settling tanks. The manager, Mr. John G. Kirchen, finds that so far there is no difference in total extraction, and apparently no great difference in cyanide consumption. It is not long since it was generally considered impracticable to cyanide ores containig sulphides. The first step was to separate the sulphides by concentration and ship them to the smelter. Later various plans of cyaniding the sulphides separately were worked out. . Then at the North Star mine the sulphides, after being concentrated and re-ground, were re-mixed

with the slime for treatment, and now Mr. Kirchen, temporarily at least, discards concentration altogether. His further results will be watched with much interest.

TREATMENT OF MILL CONCENTRATE

By R. LINDSAY

(September 30, 1911)

*As much attention has been given of late to the more careful collection and treatment of mill concentrate, a few notes on the procedure adopted at the Geldenhuis Deep, Ltd., may not be without interest. It will be understood that the following remarks apply more particularly to reduction works where operations are conducted on a fairly large scale, or to amalgamated mines working as sectional units over a reasonably limited area. The concentrate treated here results from the daily scrape from the mills at north, east, and west sections, also the big barrel residue from the respective mills; the concentrate retained on the plates with the amalgam being taken off together and transported to the central amalgam room by 11 a.m. in locked amalgam safes provided for the purpose. The scrape is then ground in barrels (a separate barrel being provided for each section) for 4 or 5 hours, and cleaned in the usual way over a stationary copper plate, the black sand settling in a tank 4 ft. 6 in. by 2 ft. by 2 ft., fitted with a baffle at 18 in. from the end adjacent to the plate, the heavy material grading 45% —200 (0.003 in.), settling behind the baffle; what remains on the farther side overflowing through a 2-in. pipe to a tank outside the building, the grading of the fine product carried over being 98% —200.

The first tank is cleaned out daily and produces about 250 lb. dry weight of concentrate, grading 45% —200 as already stated, and assaying about 30 oz. fine gold per ton. This product is then fed to a tube-mill 4 ft. 6 in. by 3 ft. 6 in., lined with 6 by 6 by 3 in. silex blocks and running at 26 r.p.m. using pebbles as the grinding medium. The product from the tube-mill, grading 98% —200, now flows into a batea running at 100 r.p.m. with a 3-in. stroke, a bath of 250 oz. of mercury being maintained therein. The batea acts most effectively as a trap, 7 oz. of mercury and 50 oz. of amalgam per ton being regularly caught. This 50 oz. of amalgam yields 11.28 oz. fine gold at a cost per ton of 3s. for power and 5s. for labor.

From the batea the concentrate pulp passes over a riffled launder and trap to the second tank, thus joining the overflow from the first tank. The overflow from the second tank passes into a third larger settling tank, the overflow from which is perfectly clear and is allowed to run to waste. The product in the last tank grades very fine, being all —200, and assays 12 to 14 oz. fine gold per ton, and represents about 5% of the total solids caught in the

*Abstract from *Jour.* Chem., Met. & Min. Soc. of S. A.

second tank. The concentrate from the big barrels in the respective mills is washed over a stationary plate in the usual way and is caught in a large tank divided into two by a partition crosswise in the centre. The product from the first compartment, amounting to 3.5 tons per month, grading 60% — 200 and assaying 7.5 oz. fine gold per ton is transported to the central amalgam room every month and is tube-milled. The product from the second compartment, amounting to 2 tons per month, grading a little over 90% — 200, and assaying 6.75 oz. fine gold per ton is transported direct to the reduction works at east section for cyanide treatment.

For cyaniding the finely ground concentrate, the conical tank designed by Andrew F. Crosse and described by him,[1] has been adopted and has proved most satisfactory. The tank, 7 ft. 6 in. diam. and 8 ft. 4 in. deep and having a content of about 122 cu. ft., treats comfortably 1800 lb. dry weight of material, and there is no trouble whatever in obtaining a clear solution for decanting, agitation with air going on meanwhile. As one would expect in treating material in so fine a state of division, the gold goes rapidly into solution, a charge assaying 250 dwt. fine gold per ton being reduced to under 30 dwt. after one hour's agitation.

The practice here is to agitate the charge with air for 12 hours before commencing to wash with 0.02% cyanide solution. At the end of 12 hours agitation the undissolved gold in the concentrate is down to 6 dwt. per ton, while the cyanide solution has a gold value of 56 dwt. per ton. Washing is then proceeded with at the rate of half a ton of solution per hour, 18% of the solution in the tank being thus displaced every hour, agitation going on meanwhile. The quantity of solution required for treating the 1800 lb. charge is 6 tons. This circulation on the Crosse principle with simultaneous decantation when washing, effects a considerable saving of time over settlement and decantation, considerably increasing the capacity of the conical tank. The washing is maintained for a period of 12 hours, making the total time of treatment 24 hours, at the end of which the concentrate residue assays 4 dwt. per ton, and the solution 0.08 dwt. per ton. This is at present the economic limit to which treatment may be carried. The cyanide consumption at 2.75 lb. of 120% KCN is rather high, due to the refractory nature of the material dealt with, but it has gradually been reduced from double the present consumption, and it is hoped that it will be still further reduced. Possibly there may be a useful field here for one of the various kinds of magnetic separators to eliminate fine iron.

The air for agitation is obtained from a little Ingersoll-Rand compressor; a ¾-in. branch pipe from it is allowed to enter a receiver fitted with a pressure gauge, just before entering the Crosse tank, thus allowing the consumption of air to be measured and the cost calculated. The consumption is 7 cu. ft. of free air per minute costing 5d. per ton of concentrate treated. The quantity of mercury recovered from the Crosse tank averages, up to the present day, only about 1 oz. per ton, thus showing the effectiveness of the

[1] *Jour.* Chem., Met. & Min. Soc. of S. A., Nov. 1909.

batea as a trap as already mentioned. The total cost of transportation, grinding, and cyaniding is 26s. per ton as follows:

		Per ton.
Tube-milling	Transport and labor	8s. 0d.
	Power	3s. 0d.
	Pebbles	4s. 0d.
	Total	15s. 0d.
Cyaniding	Transport and labor	8s. 0d.
	Cyanide	2s. 3d.
	Lime	0s. 4d.
	Air	0s. 5d.
	Total	11s. 0d.
		26s. 0d. per ton

The average value of the concentrate produced at the three sections of the Geldenhuis Deep is 16 oz. fine gold per ton, having a value of about £67. Taking the average residue at 4 dwt., and cost of treatment at 26s., the profit per ton amounts to £64 18s. The economic advantage of treating the concentrate on the mines where it is produced instead of selling it to customs works is apparent even when the basis of payment is as high as 94% of the gold content at 84s. per fine oz., less £6 per ton for treatment charges. Taking the concentrate at 16 oz. per ton as above, the profit on selling would be £56 13s. 4d., allowing 10s. per ton for bagging and transportation, the mine's profit being thus reduced by £8 4s. 8d. The concentrate plant installed here toward the end of last year at a cost of £576, including royalty on the Crosse tank, has already handsomely paid for itself, gold to the value of over £2000 having been recovered up to the end of last May.

CYANIDING TAILING AT BODIE

By An Occasional Correspondent

(November 25, 1911)

Wichman Bros., of Yerrington, Nevada, are engaged in cyaniding some old tailing, sixteen miles below Bodie on Walker river. This tailing has been exposed for a number of years, having been carried down by floods from the Bodie mills.

The method followed is to plow the tailing for ½ to 1 ft. in depth, then disk and harrow, load into wagons, and haul to tanks. As much of roots, grass, and sticks were taken out as could be easily removed. A small amount of lime is added to counteract the acidity, and solution is run on and allowed to leach, no attempt at agitation being made. A saving of about 93% is claimed. The average content was about $3 per ton. A royalty of 50c. per ton is paid, or an average of about $900 per acre, and the operators report having made a little better than wages. The small profits are due to the shallowness of the deposit. This land before the removal

of tailing would grow only a very poor quality of grass. Here is an instance of how the rancher 'gets back at' the miner, for if he collected damages from the mining company that allowed the tailing to run upon his land, then $900 per acre in royalties, and has his land in as good condition as before, he surely did not lose much. Of course, he did not necessarily collect damages.

From this point to Bodie cyaniding has been carried on over almost the entire river valley. Much of the tailing has been carried upon the land by irrigation.

CYANIDATION OF ANTIMONIAL TAILING

By W. Archer Longbottom

(June 29, 1912)

*The treatment of the refractory tailings produced from the mines of the Hillgrove district, N. S. W., has had the attention of metallurgists from time to time, but till lately the results were not satisfactory. The chief cause of the trouble is the antimony, which is present in varying quantities in all the mines. Even after careful concentration the tailing carries quite an appreciable amount. Antimony being present as stibnite (Sb_2S_3), it would be natural to expect a large consumption of cyanide, but work in the field has shown this is not necessarily the case. The cyanide used is never abnormal, even when dealing with the most antimonial tailing. The ore, as a matter of fact, is fairly high in cyanicides; stibnite, a little copper, and pyrite being present besides the gangue.

The chief difficulty is the extraction and the production of gold of good quality from the zinc precipitate. The stibnite is decomposed and the antimony gives trouble throughout the operation. It is an interesting problem to determine the actual manner in which the gold is contained in the tailing. After many experiments I have come to the conclusion that it is not in a free state. With the finest grinding, and studied under the most powerful glass, no free gold could be detected. The question then naturally arises as to how the stibnite and the gold are associated; whether the latter is contained by the former as infinitely small particles, or—and this seems more probable—whether the gold is thinly coated with the stibnite so as to prevent amalgamation. That the stibnite always contains gold can easily be proved. Assays of practically pure sulphide, even in a crystallized form, will show 8 dwt., or occasionally ½ oz. per ton.

The problem resolves itself into attacking and disintegrating the sulphide of antimony, and the freeing of the contained gold for cyanidation. The possibility that at least some of the gold is intimately associated with the stibnite is confirmed by the fact that there appears to be a limit to the value of the residue, and even the most protracted treatment will not lower it appreciably. The

*From *The Mining and Engineering Review.*

metallurgist must start, therefore, with the assumption that complete metallurgical success is practically out of the question, though a fair commercial one can be obtained. All the mines in operation on the Hillgrove field have cyanide plants working, and in addition the tailing from the Eleanora Mines, stacked in the early days, have already been treated by cyanidation—some of them twice—and are being put through again by W. H. C. Lovely, an Australian metallurgist with South African experience.

At this plant the tailing has been exposed to the atmosphere for many years, and is strongly acid. There is some difference of opinion among the local chemists as to whether tailing fresh from the battery is easier to treat than that which has been stacked long enough to permit oxidation. I believe that though the extraction may be a little lower on the raw tailing, the consumption of cyanide is much greater for an equal extraction of the old sand; also the zinc precipitate does not yield the same quality of gold as from fresh tailing. There are 18 treatment vats in use, each 20 by 4 ft., with a capacity of 30 tons. They are arranged in pairs, with a tram line running the full length of the series. They are fitted with the ordinary false bottom of slabs nailed together to conform with the circular section, and covered with hessian cloth, or, better still, cocoanut matting. Each tank has a 2-in. delivery pipe, and may be drained at will into the one common pipe leading to the gold sumps.

The sump capacity consists of six 30-ton tanks, and as quick and convenient means of pumping is provided, this sump room is ample. There are three gold tanks, each of 15 tons capacity, connected with siphons, and can be used separately at will. The precipitation boxes are six in number, and are 20-ft. long by 18 in. square. Pumping is done by 2-in. centrifugal pumps, so arranged that the material can be pumped into any sump required. There are two smelting furnaces for the clean-up, side by side, each capable of taking four '30' and two '90' plumbago crucibles, which are lifted in and out by means of an elevated lever, and are protected by clay liners. The zinc precipitate is treated with acid in six wooden tubs, 3 ft. in circumference, and there are besides six smaller vats. An ordinary flat sheet of thin iron is used for drying purposes, and, contrary to usual practice, the edges of this plate are not turned up. Power is supplied by a large Cornish boiler. The fuel used is ordinary billet wood, and about 1½ cords is burned weekly.

Method of Working.—The dump has been worked on the open-cut system, starting at the middle and working toward the edges, the sand being brought up by means of an inclined tramway. At the surface level the cars are emptied by contract into the treatment vats. The residue is disposed of also by contract. The time of treatment is seven days, including time of emptying and filling, and altogether seven washes of different strengths are run on; the strong solution first, then the weak wash, and finally a quick water wash.

The tailing has been exposed to the atmosphere for many years, the metallic sulphides have given rise to the formation of acids, and this acidity—free and latent—must be dealt with. On the whole, neutral solution, or a very slight protective alkali, is aimed at to give the most successful results. I have experimented with acid solutions, making use of the nascent hydrocyanic acid for the dissolving of the gold, and, by the addition of caustic soda, giving rise to the regeneration of free cyanide according to the equation $HCN + NaOH = NaCN + H_2O$, but the method appears to have two disadvantages: (1) the consumption of cyanide is liable to be larger with acid solutions; (2) the presence of caustic soda is troublesome, and fouls the precipitation boxes. The importance of the alkalinity or acidity of the solutions cannot be overestimated when dealing with antimonial tailing. If the protective alkali is in any excess, antimony sulphide is taken into solution and deposited on the zinc as metallic antimony, which generally means trouble in smelting.

The acidity of the tailing at Mr. Lovely's plant is neutralized by lime. This is added in calculated quantities as the treatment vat is being filled, and all precautions are taken against the solution of the antimony, at the same time allowing the sulphide to release the gold. The amount of solution made up and pumped daily on the vats is 100 tons. The solution from the treatment vats is drained away into gold sumps, and from them flows to the zinc-boxes and is passed through the precipitating room in the ordinary way, and the precipitate solution is once more made up to required cyanide strength and used again. The solution is occasionally purified by the addition of some powerful oxidizing agent, as permanganate. Lead acetate is sometimes used to accelerate precipitation, and also to throw down any sulphides there may be in solution.

The Clean Up.—At regular periods the zinc-boxes are cleaned up. The method does not differ materially from usual practice, and a few words will suffice. Only the short zinc is treated, the long zinc being replaced in the top compartments, while the lower ones are filled with new zinc. The slime is allowed to settle all night under the sulphuric acid, to make sure that all the short particles of metallic zinc has been dissolved. The supernatant liquor is then siphoned off and the heavy black slime dried over a wood fire. This slime when dried will usually average in value about 1½ to 2 oz. per pound. They are then suitably fluxed—soda, borax, silica, and flourspar being used—and smelted. Mr. Lovely departs somewhat from general practice, inasmuch as he obtains the gold in the form of base bullion, by the addition of lead oxide to the charge, reducing it by inserting rods of metallic iron in the charge. The bullion is then cupelled in specially prepared cupels, each one of a capacity of about 40 oz. This method has the advantage of giving remarkably clean ingots.

Mrs. Lovely gives the following as approximate costs of treatment.

	Per ton, s.	d.
Labor (total)	2	0
Cyanide	0	8
Zinc	0	1½
Fuel	0	2
Chemicals, etc.	0	1½
Royalty	1	0
Total	4	1

These costs are low, the cyanide item being particularly surprising considering the class of material being treated. All labor is on the contract system, and is quite satisfactory.

CYANIDATION OF CONCENTRATE

By ROBERT LINTON

(October 5, 1912)

*A series of tests recently made by me to determine the advantage or the reverse of cyaniding raw concentrate in place of shipping it to a customs smelter developed some interesting metallurgical features. The analysis of the concentrate was as follows:

	Per cent		Per cent.
Insoluble	44.00	MgO	0.47
Fe	26.50	As	0.03
Cu	1.30	S	18.20
Zn	1.20	Ag	4.29
MnO_2	0.41	Au	0.06
Al_2O_3	1.40		
CaO	1.75	Total	99.61

The silver occurred as sulphide, chiefly or perhaps wholly, in the form of argentite. The gold was in the silver sulphide. Agitation in water indicated that the concentrate was neutral, but when agitated with N/10 acid and titrated with N/10 NaOH it showed an alkalinity of 0.44%. The concentrate was ground to pass 200 mesh and the treatment carried on in a model Pachuca tank 10 in. diam. by 30 in. deep, with 1½-in. pipe for air lift and ¼-in. pipe for air feed. The first tests showed a constantly increasing alkalinity in the solutions used, although no lime was added to the charge after commencing agitation. An analysis of the solution after agitating five hours gave the following results:

	Per cent.		Per cent.
Total cyanide	0.0970	Ferrocyanides	0.0289
Free cyanide	0.0920	Sulphocyanides	0.0358
Protective alkali	0.1433	Alkaline carbonates	0.1407
Alkaline hydrates	0.0762	Zinc	0.0040
Hydrocyanic acid	0.0033	Copper	0.0688

There was no reaction for alkaline sulphides, although K_2S probably formed and immediately combined with more potassium

*From the *Jour. Chem., Met. & Min. Soc. of S. A.*, July 1912.

cyanide to form potassium thiocyanite, KCNS. This would seem to explain the increase in alkalinity observed, and to indicate that this increase in alkalinity involves a direct loss in cyanide. The reactions would be as follows:

(1) $FeS + 6KCN = K_4Fe(CN)_6 + K_2S$.
(2) $K_2S + KCN + H_2O + O = KCNS + 2KOH$.

First Test

The increasing alkalinity of the first test is shown in the following records:

Hours agitation.	Sodium cyanide added per ton concentrate, lb.	Solution KCN	Solution P. A.
2	..	0.045	0.10
2¼	20	.....	....
4	..	0.16	0.11
6	..	0.09	0.23
6¼	10	.....	....
8	..	0.195	0.207
12	..	0.115	0.22
12¼	10	.....	....
16	..	0.165	0.235
16¼	10	.....	....
20	..	0.235	0.268
20¼	10	.....	....
23	..	0.285	0.28
23½	15	.....	....
26	..	0.33	0.29
28	..	0.23	0.34
30	..	0.205	0.35
32	..	0.165	0.358

No alkali was added during the test, the increase in alkalinity resulting from the decomposition of the solutions. Several bottle tests were made to determine whether or not lead acetate would react with the K_2S and prevent the excessive consumption of cyanide. With less than 1% lead acetate, it appeared to have little if any effect; with 1% or over the increase in alkalinity above noted gradually disappeared, but at the same time the consumption of cyanide greatly increased. The results of five tests of agitating 10 gm. of concentrate with 50 c.c. solution titrating 0.73% KCN for 72 hours are as follows:

Test No.	Lead acetate added, %	KCN after 72 hr. agitation, %	P. A. after 72 hr. agitation, %
1	None	0.22	0.056
2	1	0.21	0.036
3	2	0.205	0.025
4	4	0.185	0.002
5	8	0.047	0.000

The extraction of gold and silver in each case was practically the same.

Second Test

From a study of the preliminary tests an outline of treatment was formulated and a test run of 76 hours was made in the small Pachuca agitator. Details of this run are as follows:

Hours agitation...	Cyanide added per ton conc., lb....	Lime added per ton conc., lb....	Solution. KCN, %	P.A., %	Extraction. Au, %	Ag, %
2....	....	10.9	....	....	....	....
4....	62.6	....	....	....	....	....
7....	....	....	0.58	0.31	17.4	20.8
13....	....	....	0.18	0.35	54.9	40.5
19....	....	....	0.12	0.35	79.2	52.6
22....	62.6	62.6	....	....	....	....
25....	....	....	1.30	0.36	....	....
31....	....	....	0.91	0.44	79.2	63.2
37....	....	....	0.75	0.47	....	....
43....	....	....	0.59	0.48	....	....
45....	....	....	0.56	0.48	82.6	74.6
46....	62.6	93.9	(Fresh So'n)			
48....	....	....	1.17	0.30	82.6	80.8
54....	....	....	1.08	0.35	87.3	82.2
57....	....	....	0.98	0.35	87.3	85.2
60....	....	....	0.92	0.39	88.5	87.8
62....	....	....	0.88	0.39	90.2	87.9
64....	....	....	0.87	0.39	....	....
66....	....	....	0.85	0.39	....	....
68....	....	....	0.82	0.40	90.2	89.1
70....	....	....	0.78	0.40	90.2	90.5
72....	....	....	0.78	0.40	91.1	90.6
74....	....	....	0.77	0.40	91.8	92.5
76....	....	....	0.77	0.40	92.1	92.7

Third Test

A third test was run, using less cyanide in solution, but higher protective alkalinity, and lead acetate equivalent to 3.5% of the weight of the concentrate treated, also raising the ratio of solution to concentrate to 3:1 (previous tests having been at 2.5:1). The results of this test were as given in the table on the following page, which also shows a progressive increase in protective alkalinites.

The results of all the tests run seemed to indicate that the most economical treatment was to grind the concentrate in a solution of about 0.2% KCN, then agitate in a 0.6% KCN solution, raising to about 0.8% toward the end of the treatment. Maintaining a high protective alkali proved to be of little benefit, about 0.10 to 0.15 being sufficient. The use of lead acetate appeared to be of no benefit in preventing consumption of cyanide, although the use of a small amount serves to keep the solutions clear.

Hours agitation...	Cyanide added per ton conc., lb....	Lime added per ton conc., lb....	Solution. CN, %	Solution. P.A., %	Extraction. Au, %	Extraction. Ag, %
2....	....	74	....	....	....	....
3....	23	....	0.15	0.39	....	....
4....	26	....	....	....	....	....
6....	....	....	0.37	0.44	....	....
7....	16.4	....	....	....	....	....
8....	....	....	0.27	0.55	45.0	36.8
14....	....	....	0.09	0.57	46.8	49.0
20....	....	....	0.06	0.58	54.0	68.3
22....	32.8	....	....	....	....	....
26....	....	....	0.45	0.66	55.2	68.4
32....	....	....	0.19	0.69	56.4	68.3
38....	....	....	0.17	0.70	58.1	69.2
44....	....	....	0.15	0.71	61.0	70.9
46....	....	....	0.15	0.72	63.3	71.4
47....	49.2	93.8	(Fresh	So'n)		
50....	....	....	1.0	0.39	63.8	73.5
54....	....	....	0.88	0.41	65.0	76.7
58....	....	....	0.78	0.45	67.4	77.2
62....	....	....	0.70	0.47	75.0	88.8
64....	....	....	0.67	0.48	75.0	90.6
66....	....	....	0.67	0.48	76.4	91.6
68....	....	....	0.65	0.49	80.0	91.7
70....	....	....	0.63	0.50	82.0	91.8
72....	....	....	0.62	0.50	82.0	92.2
74....	....	....	0.61	0.50	84.0	92.2
76....	....	....	0.61	0.50	85.0	92.5
78....	....	....	0.57	0.50	88.0	92.6

(November 16, 1912)

The Editor:

Sir—The article on cyanidation of concentrate by Robert Linton in your issue of October 5 recalls to mind an interesting series of tests I ran on raw concentrate at one of the Tonopah mills. With a sulphur content of 30% perfect extraction was readily obtained by cyanidation at 90°F., but the cyanide consumption was 40 lb. per ton. In testing the resultant solution it was found that KSCN was present in sufficient amount to account for about 80% of the cyanide consumption, and consequently twenty or thirty tests were run with various reagents in the attempt to prevent the formation of KSON. In repeatedly titrating for thiocyanate I was struck by the remarkable resemblance between the curves representing extraction and the concurrent KSCN titrations. After running a few tests it was possible to determine silver extraction to within a few per cent by merely titrating for K S CN. Furthermore, the sulphur present as thiocyanate was found to correspond almost exactly with the sulphur which had been combined with the

silver which had gone into solution, and until the extraction of silver became complete only the sulphur so combined enter the solution as KSCN. By prolonging the test beyond the point where all the silver was extracted, considerable sulphur, combined otherwise than with the silver, came into the solutions, so that knowing the amount of silver in the concentrate, the most economic point to stop the test could be determined merely by the KSCN titration. In extended experimental work I found nothing which prevented the formation of KSCN and finally concluded that either silver sulphide goes into solution without decomposition, or if decomposed, that the KCN snapped up the liberated sulphur ion before any preventive reagent could perform its beneficent function.

Timmins, Ontario, October 14. NOEL CUNNINGHAM.

KEEPING GOLD OUT OF CONCENTRATE

By GEORGE A. JAMES

(May 4, 1912)

A generally accepted theory of mill practice is that it is of minor importance in which part of the metallurgical process gold is recovered, if the amount does not vary, and apparent costs remain the same. The outward evidence of this belief is the elimination of amalgamating plates and amalgamation in general. It is assumed that gold that should be amalgamated will be obtained by concentration or subsequent leaching. Although this assumption can seldom be admitted, it is my purpose to show a surprising loss in net returns, even though the gold recovery does not vary in quantity. An argument advanced against amalgamation is the loss of crushing capacity owing to delays for clean-ups and dressing plates. It is my belief that amalgamators exist that overcome this objection, but if such is not the case, an extra amalgamating unit could be provided to carry the load while clean-ups are made. Admitting that both of these methods are impossible, and 10% is enough to allow for decreased crushing capacity, this would seldom add 5c. per ton to the cost of ore treatment.

I shall first indicate the advantage of amalgamation where concentration is employed and the concentrate is shipped to ore purchasers. Let us assume ore worth $20 per ton and that 10 tons have been concentrated into 1. The ton of concentrate would contain $200. The percentages usually deducted would be 8.2 (5% of assay value and gold at $20 per ounce), or $16.10 per ton of concentrate, equal to $1.61 per ton of original ore. In addition to this, there are various charges not so apparent, but which amount to considerable importance; such as leakage from sacks in shipment and portions adhering to emptied sacks. Such leakage will certainly come to an average of 3 lb. per sack. Allowing 15 sacks

to ship the ton of concentrate, 451 lb. would thus be lost, which at 9c. per pound would represent a loss of $3.95. In grinding samples in disk pulverizers (as the custom is to reduce to 100-mesh or finer), ½% or more of iron enters the sample, which, of course, deducts that much from the assay value. The commercial assay is usually ½% below corrected assays, and in determining moisture the buyer always 'plays safe.'

Considering these details, it will be seen that the ton of concentrate has returned $178.60, not including freight and treatment charges, or 89.3% per ton of original ore, equal to a treatment cost of $2.14. Let us assume that 50% of the gold could be recovered by amalgamation and that the extra cost of treatment is offset by factors not included in estimating the expense of shipping the concentrate to the reduction works. Under this head might be mentioned extra control assays, increased freight charges (which are usually regulated according to the value of ore shipped), interest on money locked up in settlements. The expense charge is about 5c. per ton for refining of gold and expressage. The account under these circumstances would then stand:

Amalgamated gold, net	$ 99.50
Gold shipped	89.30
Total	$188.30

Or $1.02 per ton in favor of this manner of handling. Against this should be charged the interest on investment required to obtain equal capacities.

An example of cyanide or chlorination with the same ore would be approximately as follows. It is not too much to assert that the gold most easily amalgamated is least susceptible to leaching methods (as it consists of the coarser particles), and where 50% could be amalgamated, total extraction would be reduced, at least 2%, and the time of treatment necessarily increased. Such being the case, the comparison would then be:

By leaching:

50%, with 97% extraction	$ 97
50%, with 90% extraction	90
Total net return	$187

By amalgamation:*

50%, with 100% extraction	$100
50%, with 97% extraction	97
Total net return	$197

Thus showing a profit of $1 per ton of original ore treated. This article is merely intended to attract attention to the principle involved, and to a common source of error.

*It is assumed that refining and express costs remain the same in case of gold recovered by both amalgamation and leaching, although it is usually in favor of amalgamation.

CYANIDATION OF PYRITIC ORE

By F. B. Reece

(July 20, 1912)

The cyanidation of concentrates and of ores of high sulphide content, has been described, and the difficulties attendant discussed, by J. W. Hutchinson, A. B. Parsons, Huntington Adams, and others.

Some heavy sulphide ores which do not offer any special difficulties as regards excessive consumption of chemicals, and which yield a satisfactory extraction to cyanidation, give much trouble from a mechanical point of view. I was called upon to operate a plant, situated in a remote district, which had been designed and erected by others to treat an ore of this character, a mixture of arsenopyrite, pyrite, galena, and blende. The equipment consisted of stamps, tube-mills, cone-classifiers and thickeners, 'Brown' type agitators, and vacuum leaf-filters. Supplies and renewals took from four to six months from date of order to reach the mine. The valuable part of the ore was the sulphide, and to maintain the ore at a profitable grade it was necessary to sort it so that it contained from 20 to 25% of sulphides. Due to the necessity of using very soft local pebbles the regrinding was not as fine as is desirable for the proper agitation of such material. On an average 90% passed 200 mesh.

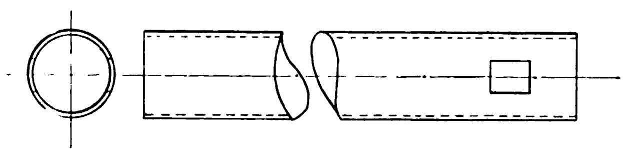
The supply of air for the four 27½ by 10-ft. tanks was inadequate. Allowing for the altitude, 7000 ft., it amounted to a total of 32 cu. ft. of free air per minute at 25 lb. maximum pressure. It was found impossible to do more than agitate two tanks at one time, and continuous agitation was out of the question until additional capacity in air could be provided. The design of the central agitation pipe was faulty. It was originally 16 inches in diameter and was far too large, and set so low that there was only a 3-in. annular space between the bottom of the pipe and sides of the cone of tank. This space would probably be too small to prevent choking while agitating a purely silicious slime, it was found to be quite inadequate for the material under discussion. After long and expensive delays and when two or three charges had been lost, the centre pipes were replaced by others of 11 in. diameter placed so that their lower ends were 22 in. from the apex of cone, giving a clearance of about 7 in. between pipe and side of cone.

Bernard MacDonald has drawn attention to the necessity of using the smallest air-lift tube consistent with the amount of liquid to be pumped. His opinions were certainly justified in this case. Smaller pipes than 11 in. would probably have been advantageous, but none were available.

Further, the suggestion put forward by Lloyd Kniffin of reducing the length of the central pipe, was carried out, leaving the pipe about two-thirds of its original length. These alterations caused a great improvement in the agitation. To assist in keeping

the solid matter in agitation during the time of filling and emptying, two holes were cut in the lower part of centre pipe on opposite sides of pipe. The tendency to settle was much reduced by this simple expedient, and it was decided to cut two more holes about 8 or 9 ft. higher up, but unfortunately no opportunity was afforded for trying this experiment. The effect of these apertures on the circulation of pulp was very noticeable as the level of pulp gradually rose during filling of a tank. It is questionable if they are an advantage after the tank is in full agitation, but this could easily be overcome in the manner suggested by Mr. Adams at the Natividad mill, where sliding doors were fitted on the central air-lift tube and opened and closed as the level of pulp in tank varied.

Much trouble was experienced from the choking of the ¾-in. air-agitation pipes which had not been provided with non-return valves. These pipes were the only means that had been provided for agitation, the usual side and 'spider' pipes not having been supplied. It was impossible to install these during the time of running covered by this article. The only remedy was to remove, clean, and replace this pipe after every stoppage of the air supply.



IMPROVED AGITATION PIPE

To force this down again into the mass of settled slime the end of pipe was closed and drawn out to a sharp point. Two sets of ⅛-in. holes were then drilled through the pipe, the lower set coming very close to the bottom of cone when the point of pipe rested on the apex, and the other set coming 3 in. above the intake of the central column.

Discharging the tanks to pulp-stock tank by gravity was a tedious process on account of the lack of means of efficiently agitating the charge to the last moment. The leaf-vacuum-filter employed gave poor results, as might have been expected in treating material of this weight. It was impossible to prevent segregation and dropping of the bottom and heavier portion of cake. The only kind of filter to use in a case like this is a pressure-filter, making cake very rapidly, and preferably one which has no excess slime to displace. The solution of the difficulty in a case like the foregoing would appear to be the employment of mechanically driven classifiers and thickeners, of continuous agitation, pressure-filters, and a plentiful supply of air to the agitation and pulp-stock tanks. I am convinced from personal experience at the Hacienda Guadalupe, Pachuca, that it is not necessary to use the 'diagonal' method of connecting the tanks in series recommended by Mr. Grothe and Mr. Kuryla. The overflow method

used by Mr. Adams gives excellent results, with more simple construction.

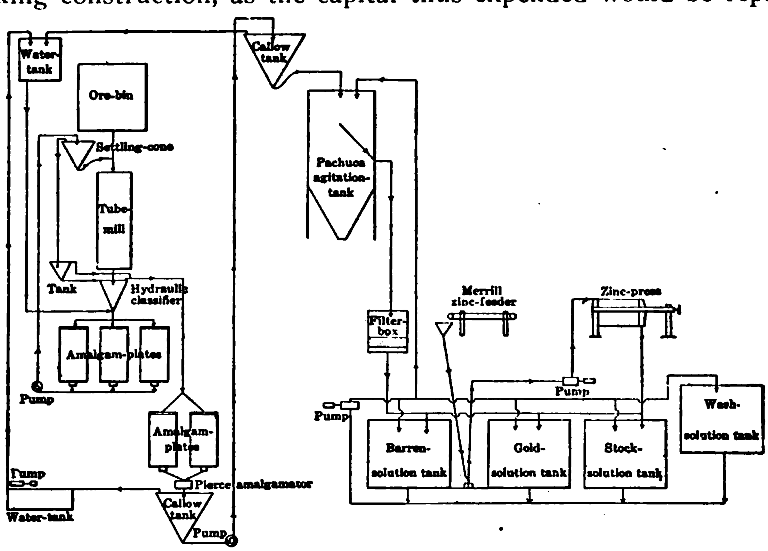
The foregoing experience shows how necessary it is to carefully design agitation plants to deal with material containing more than say 6 or 8% of sulphides.

W. P. Lass* has described and illustrated a valuable device in the shape of an adjustable spider pipe, which can be lowered on top of a settled charge, to gradually bring it into a state of agitation. In agitating this heavy material, it is a great advantage to be able to remove the air pipes at will, as, despite all care that can be taken, choke-ups are not infrequent. Furthermore, a pipe which is in its correct position when the tank is in brisk agitation, may be quite unable to make headway against the depth of settled pulp which overlies it, when a tank has settled for more than a few minutes. It will be noticed in referring to the article by Mr. Lass that the central air-lift pipes are 10 inches in diameter for a 10-ft. diameter tank. Probably a still smaller pipe would be even more effective.

AN EXPERIMENTAL CYANIDE PLANT

(November 18, 1911)

In designing a cyanide plant at the Alaska Treadwell mines, it was judged wisest to make exhaustive experiments before undertaking construction, as the capital thus expended would be repaid



FLOW-SHEET OF EXPERIMENTAL CYANIDE PLANT

in the more effective working of the mill as finally constructed, and by the saving in the cost of the alterations usually necessary in a new plant. Preliminary tests were made, under the direction of

*_Mining and Scientific Press_, Oct. 21, 1911. (Reprinted Page 273, this volume.)

W. P. Lass, in glass jars, and followed by tests upon 50-lb. lots by grinding in a small clean-up barrel filled with iron balls, the pulp being agitated in a 14 in. by 4-ft. Pachuca tank. Next a 4 by 12-ft. Abbé tube-mill was installed, the pulp passing over amalgamated plates to a Dorr classifier and Callow cones were used for dewatering the sand before cyaniding in a 10 by 22-ft. Pachuca tank, the classified solution going to a Merrill precipitation press. With this equipment 15-ton lots could be treated. The gold content was removed from the pulp by successive washes and treatment. The accompanying flow-sheet shows diagrammatically the method used for these experiments, with the exception that the filter-box shown was later superseded by a Kelly filter-press (type 1 B) of 50 tons daily capacity, which did away with the numerous washes and decantations previously required. With this equipment a complete series of tests was carried out, with the result that when the 100-ton plant was constructed it was operated from the start without any hitch.

CYANIDE PLANT AT THE TREADWELL MINES, ALASKA

(Editorial, October 21, 1911)

Cyanidation offers problems as many faced as a diamond, and one of the sides that recently has been to the fore presents the question involved in the cyanidation of concentrate. In his well written and interesting paper, which we print in abstract this week, Mr. W. P. Lass describes the careful experimental investigations carried on at the Alaska Treadwell and the final successful carrying out of the operation, and the many ingenious modifications introduced to meet the special conditions there existing will be of much interest to our readers. It is interesting to note further that in the beginning of the cyanide process the development of the chemical theory was far in advance of its mechanical development. But in recent years the many and varied mechanical problems have been attacked with so much energy and ingenuity that there is now more likelihood of the mechanical equipment being in advance of the chemical development of the process. The chemical side has recently been touched upon by Mr. J. E. Rothwell, who calls attention to the fact that the process is essentially a chemical one, and that mechanical equipment, though enormously important, is in reality subordinate to the chemical reactions involved. It is safe to predict that the most notable steps in the advance of the application of the cyanide process during the next few years are likely to be chemical rather than mechanical. We are accustomed to look to the technical schools and universities for the theoretical development of technical subjects, but in this respect the results have of late been singularly disappointing. One of the exceptions is a notable paper of slime filtration by Mr. G. J. Young, which we will present in abstract next week.

CYANIDE PLANT AT THE TREADWELL MINES, ALASKA

By W. P. Lass

(October 21, 1911)

*The purpose of this article is not only to describe the plant and method of cyaniding the Treadwell concentrate, but to present some of the results of experimental work during the past three years for the Alaska Treadwell Gold Mining Co., at Douglas Island, Alaska, under the direction of F. W. Bradley, consulting engineer, and Robert A. Kinzie, general superintendent, of the affiliated companies.

At the time the experimental work was undertaken the concentrate was being shipped to the smelter at Tacoma, and the cost for treatment of 3-oz. (gold) concentrate was $11.95 per ton.

From the experimental work it was estimated that 96% extraction could be made by treatment at the plant, and that the cost, when treating 80 tons per day, would be $3.25 per ton. Adding to this the 4% treatment loss, which on 3-oz. concentrate amounts to $2.48, gives a total cost of $5.73 per ton, a net gain of $6.22 per ton by the local treatment. In addition to this saving, the cyanide tailing would have an economic value due to the sulphur and iron content, as well as the value of the residual gold after oxidation.

The concentrate, amounting to 1.8% of the original ore, contains: Fe, 40%; S, 1%; SiO_2, 11%, and carries from 2.5 to 4 oz. gold and 0.75 oz. silver per ton. The gold and silver amount to about 37% of the original content. The figures in the following table are assays and averages of sizing tests on concentrate from the various mills.

ASSAY SIZING TESTS OF TREADWELL GOLD AND SILVER ORES

Size of material.	Wt. %	Assay value per ton.	Value %	Value in 1 ton of original.
On 20-mesh screen......	0.44	$ 70.35	0.48	$ 0.31
Through 20, on 40....	8 23	203.96	26.05	16.83
Through 40, on 60....	10.96	143.89	24.30	15.76
Through 60, on 80....	12.49	94.88	18.34	11.85
Through 80, on 100....	10.38	60.85	9.78	6.32
Through 100, on 120....	13.37	39.27	8.14	5.25
Through 120, on 150....	7.69	26.61	3.17	2.05
Through 150	36.46	17.10	9.65	6.23
	100.00		100.00	$64.60
On 20-mesh screen......	044	$70.35	0.48	$0.31

Preliminary tests were made in glass jars, after which 50-lb. composite samples were ground to 200 mesh in a clean-up barrel provided with iron balls, the pulp passed over amalgamated plates and then agitated in small Pachuca tanks, 4 ft. high and 14 in. diameter. The results showed that 75% of the gold could be re-

*Abstract of a paper presented at the San Francisco meeting of the American Institute of Mining Engineers.

No. of Test.	Tons Treated.	* Assay Per Ton. Original Concentrates.	* Assay Per Ton. Ground Product.	Treatment During Grinding.	Extracted During Grinding.	Ground Product Through 200-Mesh.	Ratio Ore to Solution.	* Assay Per Ton. Agitation-Heads.	* Assay Per Ton. Cyanide-Tails.	Extracted by Cyanide in Pachuca.	Cyanide Solution Used.	Lead Acetate Per Ton Concentrates.	Time of Agitation.	Cyanide-Loss Per Ton Concentrates.	Changes of Solution.	Total Extraction.
					Per Cent.	Per Cent.				Per Cent.	Per Cent.	Pounds.	Hours.	Pounds.	No.	Per Cent.
1	20	$42.16	$19.63	Amalgamation	54.0	94.0	1 to 1.6	$19.63	$4.00	79.0	0.35	....	48	6.70	2	90.0
2	15	54.80	9.00	Amalgamation	83.0	99.8	1 to 2.2	9.00	.96	89.0	0.15	1.5	18	4.00	3	98.3
3	18	42.00	10.40	Amalgamation	75.2	99.6	1 to 2	10.40	1.40	86.4	0.125	1.14	16	2.97	3	96.6
4	16	47.20	10.00	Amalgamation	78.8	99.8	1 to 2	10.00	1.20	88.0	0.1	1.0	16	1.92	3	97.8
5	10	32.00	8.40	Amalgamation	73.7	97.0	1 to 3	8.40	.80	90.4	0.14	1.0	24	2.00	2	97.5
6	10	56.80	9.60	Amalgamation	83.1	97.0	1 to 3	9.60	1.80	81.2	0.07	....	10	2.30	3	96.8
8	8	50.00	8.80	Amalgamation	82.4	98.0	1 to 3	8.80	2.30	73.8	0.038	1.0	18	1.44	2	95.4
9	9	30.80	9.00	Amalgamation	70.7	97.0	1 to 2	9.00	1.60	71.7	0.1	....	12	1.08	2	94.8
10	15	52.00	14.60	Amalgamation	71.9	90.0	1 to 2.1	14.60	2.00	87.6	0.15	1.0	24	0.90	2	96.1
11	15	37.20	13.40	Amalgamation 0.002 per cent. cyanide.	64.0	88.0	1 to 2	13.40	2.40	82.0	0.15	1.0	24	0.96	2	93.6
12	15	40.80	9.60	Amalgamation 0.05 per cent. cyanide.	76.5	98.0	1 to 2	9.60	2.40	75.0	0.15	....	24	4.46	2	94.0
13	15	46.00	9.40	Amalgamation 0.025 per cent. cyanide.	79.6	100.0	1 to 2	9.40	1.70	82.0	0.15	0.67	24	2.20	2	96.3
14	15	36.80	9.80	Amalgamation 0.05 per cent. cyanide.	73.4	100.0	1 to 2	9.80	1.60	88.6	0.15	....	24	3.50	2	95.0
15	15	36.00	5.00	0.07 per cent. cyanide.	86.1	100.0	1 to 2	5.00	1.12	77.6	0.15	1.0	24	4.40	2	96.9
16	7	48.00	18.00	Concentration	62.4	100.0	1 to 4	18.00	.90	95.0	0.15	....	18	1.60	2	98.5
17	7	48.00	16.90	Concentration	64.7	100.0	1 to 4	16.90	1.10	93.5	0.15	....	18	1.86	3	97.6
18	9	48.00	19.20	Concentration	60.0	99.0	1 to 3	19.20	1.80	90.6	0.075	....	18	2.28	3	96.2
19	12	48.00	13.20	0.05 per cent. cyanide.	72.4	99.0	1 to 2	13.20	2.60	80.8	0.1	....	12	5.10	2	94.6
20	10	48.00	12.00	.046 per cent. cyanide	75.0	99.0	1 to 2.2	12.00	2.60	78.8	0.033	....	18	4.80	3	94.5
21	10	32.00	14.00	.018 per cent. cyanide	56.2	99.0	1 to 2.3	14.00	2.80	80.0	0.036	....	36	3.70	4	91.2
22	18	28.00	22.40	.024 per cent. cyanide	20.0	97.0	1 to 2	22.40	2.40	89.2	0.025	....	24	2.90	3	91.4
23	16	33.00	12.20	.032 per cent. cyanide	63.0	99.0	1 to 2	12.20	.90	92.6	0.125	....	24	4.70	3	97.2
24	16	55.00	15.35	.028 per cent. cyanide	72.1	99.0	1 to 2	15.35	1.60	89.5	0.12	....	16	5.20	3	97.8
25	15	46.00	17.60	.042 per cent. cyanide	61.8	99.0	1 to 2	17.60	2.08	88.2	0.12	....	16	5.00	2	95.4
26	15	55.00	8.00	0.066 per cent. cyanide	85.4	99.0	1 to 2	8.00	2.90	63.7	0.12	....	16	4.30	2	94.8

* All values are in gold at $20 per ounce. Tests 2, 6, 8, 9, 10, 16 and 17 given preliminary alkali agitation.
All solutions contained an excess of lime.

RESULTS OF FIRST 25 EXPERIMENTS ON CYANIDING CONCENTRATE.

covered by fine grinding and amalgamation and 96% by amalgamating followed by cyaniding. The next step was the construction of an addition to one of the mills and the erection of a 4 by 12-ft. Abbé tube-mill. Various forms of classifier were tried, the Dorr proving most satisfactory. Callow cones were used for dewatering, and the agitation was carried on in a Pachuca tank 10 ft. diam. and 22 ft. high. The complete results of the first 25 experimental runs are seen in the table given below. Throughout this paper the ton of 2000 lb. is used, with gold at $20.67 per oz.; no account is taken of the value of the silver.

The results of the tests showed that 75% of the golld could be recovered by grinding and amalgamating, or 96% by the combined method of amalgamating and cyaniding. Results also showed that during the process of grinding in 1.5-lb. (0.075%) cyanide solution, a similar extraction could be obtained without amalgamation. Thus a satisfactory extraction was obtained either by amalgamating and cyaniding or by cyaniding direct. A preliminary agitation with an alkali solution was found to shorten the time of cyanide treatment and save 25% in the cyanide consumption. Passing the air used for agitation through a receiver filled with a solution of caustic soda or milk of lime also decreased the cyanide consumption, presumably by the removal of oil and CO_2 from the air. When grinding in solution over 1 lb. per ton (0.05%), followed by amalgamation, it was difficult to keep the plates bright, due to a dull white surface deposit, which if allowed to remain turned to a dull gray. A muntz-metal plate was substituted for a copper plate, and as all the plates were coated no variation in the result was noted, and there did not appear to be any essential advantage gained from its use.

The results obtained from this extended period of investigation, lasting over two years and costing $27,794, justified the building of a plant of 100 tons daily capacity. The preliminary cost was largely offset by the ability of the final plant to treat the concentrate without the usual alterations necessary in starting a new mill. It also produced the nucleus of the final mill crew. As the abandonment of amalgamation in favor of direct cyaniding seemed a somewhat radical change, the new mill was planned to operate either way, and ultimately nearly 5000 tons was treated by each method before deciding to cast out the time-honored amalgamated plate. All of the equipment purchased for the experimental work was used in the permanent plant, which was completed in September, 1910.

The cyanide plant consists of three buildings situated on a hillside 200 ft. above the stamp-mill. The upper building contains the grinding and amalgamating plant, with a lower floor for solution-storage tanks. The lower contains the cyanide equipment proper, while the refinery is in a concrete building at one side, as shown in the illustration. The five mills on Douglas island contain a total of 900 stamps, and crush approximately 5000 tons per day. The crushed ore after amalgamation is concentrated on 360 Frue

vanners, yielding an average of 90 tons of concentrate per day, of from 2.5 to 4 oz. gold per ton. The flow-sheet of the operations is given on the next page.

From the vanner-boxes the concentrate is shoveled into specially constructed flat-bottomed steel cars. These cars, each holding two tons of concentrate, are made up into trains at the mills, and brought by locomotives to the foot of the incline below the cyanide plant. This incline is 900 ft. long with 14° rise. A Union Iron Works geared hoist, driven by a 75-hp. electric motor, brings the train to a switch above the upper building. Beginning with this switch, the entire plant is in duplicate throughout. Leaving the switch by gravity, the cars are weighed, sampled, and run into revolving tipples. Upon releasing the brake the tipple revolves, turning the car bottom up and dropping the load from the car. The change in the centre of gravity then causes the tipple to right itself, and the empty car is weighed and returned to the main switch.

Most of the water is removed from the concentrate while in the vanner-boxes by the aid of a bumper, which is simply a large air-piston machine mounted on a truck and moved from box to box. The bumping causes the particles to readjust themselves and pack in the bottom of the box, while the water runs off, leaving about 12% of moisture in the concentrate. It is considerably easier to shovel the concentrate into the cars after the bumping. The concentrate is sampled while in the car by means of a long ship-augur. With the ordinary long spoon it was impossible to obtain satisfactory checks in the samples, as the concentrate is usually covered with water. Unslaked lime is added to each of the empty cars as it leaves the tipple in order to reach the concentrate at the earliest possible stage. It also forms a line of cleavage, causing the concentrate to dump clean from the bottom.

From the cars the concentrate falls into 100-ton steel storage-bins, 15 ft. diam., with 55° conical bottoms. The concentrate in the bins is kept covered with water, which effectually prevents oxidation of the sulphides while lying in the bins. From this point until the cyanide treatment begins the concentrate is in strong lime solution at all times. At the apex of the conical bottom of each bin tight-fitting gates control the outflow, which is at once sluiced directly into Dorr classifiers. The sluicing medium is the coarse return product referred to later. There are three Dorr classifiers driven by one 7.5-hp. electric motor, one feeding into each tube-mill and making 24 strokes per minute. This rate of speed, causing greater agitation, was found necessary to separate the large bulk of the fine from the coarse.

The coarse product of the classifiers falls into the spiral feeders of the tube-mills. These mills are of the Abbé type, 5 by 22 ft., lap-welded, trunnion bearings, with corrugated sectional liners; 3-in. Danish flint pebbles are used for the grinding. Two 75-hp. motors on three-phase circuit at 2200 volts are belted to an overhead central line-shaft, which in turn is belted to the pinion shaft

of the tube-mills. The tubes are driven from the discharge ends and make 27 rev. per min. The mills are controlled by friction-clutch pulleys on the central line-shaft. For the period from May 15 to July 15, 1911, one tube-mill ground at the rate of 88.75 tons of concentrate per 24 hours actual running time, the power consumption averaging 64 hp. By replacing each 75-hp. motor with a 100-hp. motor, and substituting leather for canvas belts on the main drive, the power consumption was reduced to an average of 50 hp. for the same tube duty. This was with the tube just half filled with pebbles, the normal running load. By increasing the pebble load to 6 in. above the centre of the tube, the power consumption rises to 75 hp., and both the quantity of tube feed and the fineness of the product discharged are increased.

The following is an average screen analysis of the feed and discharge of one 5 by 22-ft. mill, when grinding an original feed of 88.75 tons per 24 hours:

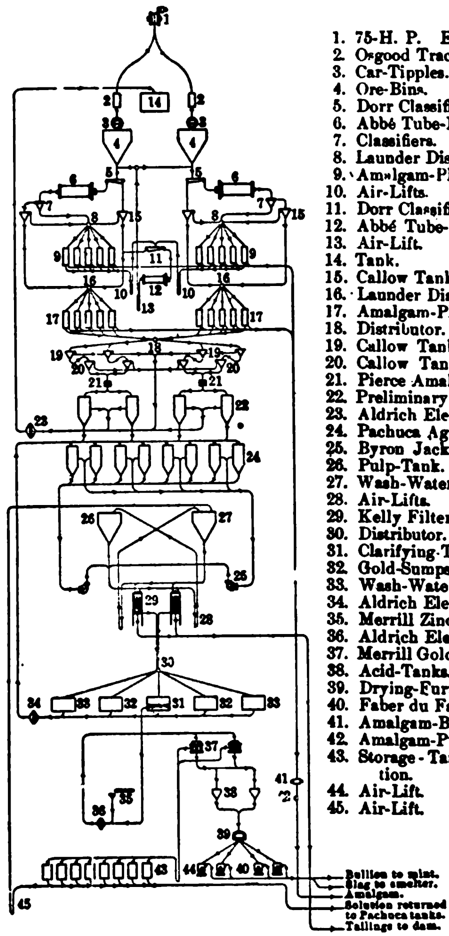
	On 100 mesh.	On 200 mesh.	Through 200 mesh.
Feed	48.7	41.5	9.8
Discharge	10.1	26.4	63.5

The pulp contained 38.5% moisture.

When the concentrate is amalgamated previous to cyaniding, the product discharged from the tubes is distributed over 10 copper amalgamating plates, each 4 ft. 8 in. wide by 10 ft. long plated with 2 oz. of silver per square foot. The pulp flows from the plates into launders built into the floor. No traps are used, as they are quickly clogged by the metallic iron which accumulates in the concentrate from the wear of the various machines used in the processes of mining and milling. This iron, if allowed to accumulate in the coarse return-product, will amount to as much as 15% of the total. Experiments are now being carried on with a magnetic device for removing the iron from the pulp. From a sump in the launder an air-lift elevates the pulp to a spitzlutte, from which the coarse material is continuously drawn into a Dorr classifier, the coarse from which feeds a 4 by 12-ft. Abbé tube-mill, similar to the larger ones described above. The discharge from this mill joins the overflow from the spitzlutte, and is elevated by air-lifts to two settling-cones, so situated that the spigot-discharge from them becomes the sluicing medium for the original feed referred to above.

Two points will be observed here: (1) that the Dorr classifiers are at present doing all the classifying for the mill; and (2) that the concentrate is carried around in a closed circuit from which there is no escape until the particles have become fine enough to join the overflow from the back of the Dorr classifiers. The Dorr overflow, which is the product cyanided, is more than 98% through 200-mesh. The remaining 2% is silica from the wear of the pebbles. Of the concentrate, the entire product will pass a 200-mesh screen. The overflow from the Dorr classifiers passes into two Callow dewatering cones, the spigot product of which is distributed over 10 amalgamating plates similar to the coarse amalgamating

plates previously described. From the plates the pulp flows into launders, thence into a 6-in. pipe, 37 ft. long, having a fall of ¾ in. per foot, which conveys the pulp directly to the lower or cyanide building. In the lower building the pulp is received into a wooden distributing-box, from which it flows through two Pierce amalgamators into four 8-ft. Callow cones. The spigot-product from these

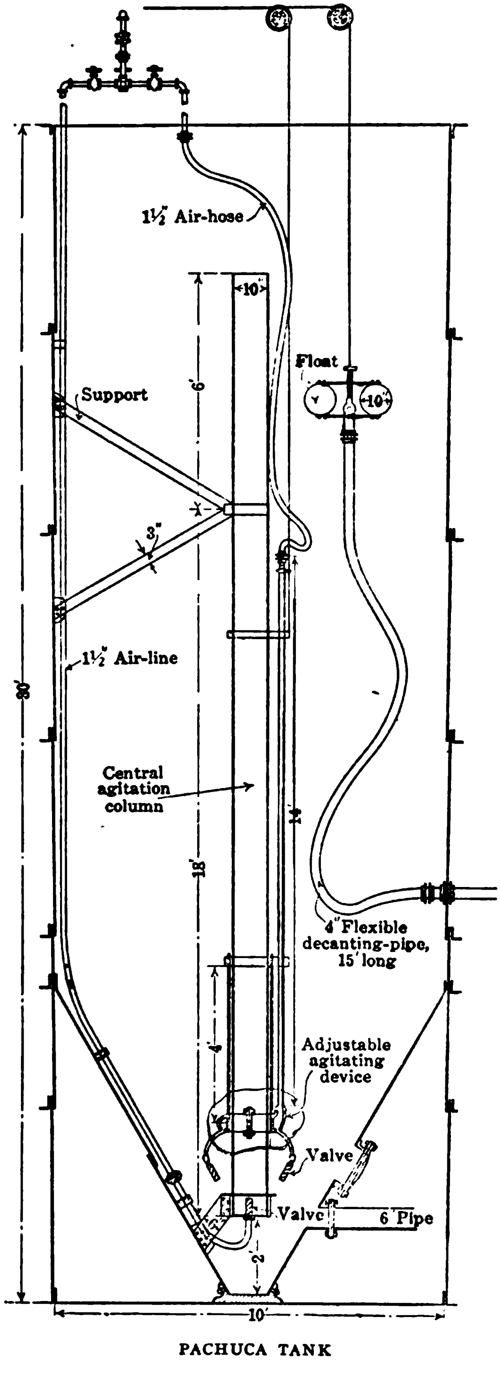


1. 75-H. P. Electric Hoist
2. Osgood Track-Scales.
3. Car-Tipples.
4. Ore-Bins.
5. Dorr Classifiers.
6. Abbé Tube-Mills.
7. Classifiers.
8. Launder Distributors.
9. Amalgam-Plates for coarse pulp.
10. Air-Lifts.
11. Dorr Classifier.
12. Abbé Tube-Mill.
13. Air-Lift.
14. Tank.
15. Callow Tanks.
16. Launder Distributors.
17. Amalgam-Plates for fine pulp.
18. Distributor.
19. Callow Tanks.
20. Callow Tanks.
21. Pierce Amalgamators.
22. Preliminary Agitation-Tanks.
23. Aldrich Electric Triplex Pump.
24. Pachuca Agitation-Tanks.
25. Byron Jackson 4-in. Cent. Pumps.
26. Pulp-Tank.
27. Wash-Water Tank.
28. Air-Lifts.
29. Kelly Filter-Presses.
30. Distributor.
31. Clarifying-Tank.
32. Gold-Sumps.
33. Wash-Water Sumps.
34. Aldrich Electric Triplex Pump.
35. Merrill Zinc-Feeder.
36. Aldrich Electric Triplex Pump.
37. Merrill Gold-Precipitation Presses.
38. Acid-Tanks.
39. Drying-Furnace for Precipitate.
40. Faber du Faur Furnaces.
41. Amalgam-Barrel.
42. Amalgam-Press.
43. Storage-Tanks for barren solution.
44. Air-Lift.
45. Air-Lift.

FLOW-SHEET, 100-TON MILL

cones discharges into four similar ones placed lower than the first set.

The spigot-product from the lower cones enters one of four Pachuca tanks, where it receives a preliminary treatment of 3 hr. agitation in a solution containing 2 lb. of lime per ton (0.1%), after which it is allowed to settle and the clear solution is decanted. The filling, agitating, settling, decanting, and discharging of a 25-ton charge of concentrate, which includes 46 tons of lime solution,



PACHUCA TANK

requires somewhat less than 24 hr. This preliminary treatment saves in the subsequent treatment at least 1 lb. of cyanide per ton of concentrate. The overflow lime-water from the Callow cones enters the same sump with the decanted lime-water from the preliminary treatment, and is pumped by an Aldrich triplex 7 by 9-in. electric pump into a reservoir of 75 tons capacity situated in the upper building. The thickened pulp, ranging from 1.8 to 2.2 specific gravity, is drawn into one of eight Pachuca tanks, where it is given the cyanide treatment. All Pachuca tanks in the mill are 10 ft. in diameter and 30 ft. high, with 60° conical bottoms. When filled to the level found best for agitating (which is 6 in. below the top of the central column), each tank holds a volume equivalent to 50 tons of water. This is equal to the regular charge of 30 tons of concentrate with 40 tons of solution, although as high as 40 tons of concentrate has been treated as one charge without any difference in extraction results. The floors under the Pachuca tanks, as well as all other floors in the building, are of smooth concrete, sloping to a central sump, supplied with small pumps to return any escaped solution or pump to the proper tanks.

The first cyanide treatment consists of 8 hr. agitation in a 2-lb. (0.1%) cyanide solution; either potassium or the mixed cyanides being successfully used. Alkali is kept at 1.25 lb. (0.063%) of lime (CaO) per ton of solution. Lime is added during the treatment if the titrations show below that figure; 18 hr. is allowed for settlement and decantation of this solution. Decantation takes place through a flexible hose, which is made as follows: Canvas coated with tar is wrapped around pieces of old boiler tubing 3 in. diameter and 4 in. long, spaced ¾ in. apart. The canvas between the short lengths of tubing is wrapped with wire, making the diameter of these spaces slightly smaller than that of the tubing, thus insuring flexibility as well as avoiding the shifting of the tubing. Attached horizontally to the top of the flexible hose is a 3-in. slotted pipe. In operation this slotted intake floats by the aid of two adjustable air-cylinders. The arrangement of these cylinders is such as to allow the vertical adjustment of the intake-pipe to any depth of submergence desired.

The long settlement allowed, with the excessively fine condition of the concentrate, its high specific gravity, from 4.6 to 5.0, and the high alkalinity of the solution, leaves a 30-ton packed mass in the bottom of the Pachuca. This is brought into agitation within 15 min. by a device designated as the 'spider,' which is an adjustable hollow annular casting with radiating fingers, the whole encircling the central agitation-column. When the charge is to be put into agitation the spider is lowered by a small hand-windlass until it rests on top of the settled charge. Air is then turned through the fingers, and at the same time the solution for the next treatment is run into the tank. The device rapidly bores its way to the bottom of the Pachuca, leaving a boiling, churning pulp above, and clearing the way to the bottom opening of the central 10-in. agitating-column. As soon as this is opened and air has been admitted to the inner pipe, the spider is raised from the tank and full agitation of the charge proceeds.

The second cyanide treatment of the charge is with solution drawn from the barren-solution storage-tanks or the wash-solution storage, the cyanide strength being 1.5 lb. (0.075%) per ton of solution. After 2 hr. agitation the air is shut off and almost immediately decantation is started. This decanted solution is pumped directly on to an incoming fresh charge, being strengthened in cyanide as it enters the tank, and becoming the first cyanide solution for the new charge. This cycle in handling solution—barren to wash-solution, then to second cyanide treatment at 0.075% cyanide, then to first treatment at 0.1% cyanide, thence to precipitation and back to barren—gives to each step just the conditions best suited for that step, and is very satisfactory in practical operation. The settled pulp after the second decantation has a specific gravity of 1.8, and is readily agitated by means of the spider, and then discharged into the pulp-storage tank by a Byron Jackson 4-in. centrifugal pump, from which it is drawn to the Kelly filter-press. This thick pulp holds in suspension the sand which would settle

through a lighter medium. The storage-tank is conical bottomed, 15 ft. in diameter, and situated at such an elevation that a static pressure of 30 lb. per square inch is exerted at the filter-presses. The pulp in the tank is kept in constant circulation by an air-lift, drawing from the conical bottom and carrying the pulp down close under the filter-presses and back up again over the top of the tank. The pulp as pumped from the Pachuca tanks enters the bottom of this same line, and the whole is thus kept in suspension and circulation past the presses, into which it is intermittently drawn for filter-treatment. Above the pulp-storage tank is placed a similar tank for the storage of wash-water, from which a hydrostatic pressure of 25 lb. per square inch is obtained at the presses. This solution is kept in circulation, using the same method as applied to the pulp. The higher gravity of the pulp in the lower tank results in a greater pressure at the presses than that obtained from the wash-solution, although the latter carries a higher head.

Filtering is done in two type 1-B Kelly presses. By opening valves in the circulation-lines directly under each press it is filled with either pulp or wash-solution as desired. The excess pulp or wash-solution from the press-cylinder is returned into its proper line by displacing with compressed air admitted into the cylinder. The amount of wash given depends upon the comminution of the concentrate, the usual pulp being washed with 0.5 ton of solution per ton of concentrate. The cake formed during decantation of the first treatment-solution, being very fine slime and more impervious to wash-solution than the regular pulp, is given 1 ton of wash per ton of concentrate. When filling the press, the contained air is allowed to escape through an overhead pipe attached to the highest point of the press-cylinder. The change in sound of the exhaust indicates to the pressman when the press is full. After drying the cake with compressed air until it contains not more than 10% of moisture, the press is opened and the cake shaken off with wooden paddles, and then sluiced with water to the tailing-dam. A distributor below the press-launder sends the gold-solution to two gold-sumps and the wash-solution to the two wash-solution storage-tanks. These four tanks, as well as a clarifying-tank which is in the same group, are built of 3-in. redwood, 15 ft. in diameter by 16 ft. deep, and each holds 75 tons of solution.

The wash-solution is pumped to a Pachuca tank as needed, becoming a second-treatment solution. From the gold-tank the solution is drawn into the clarifying-tank, in which are suspended vertically six canvas filter-leaves, all connected to the suction of a triplex 7 by 9-in. Aldrich electirc pump, used exclusively for pumping gold-solution through the precipitation-presses. A traveling-belt, driven by ratchet-gears and a pair of eccentrics connected to the pump-drive, feeds zinc dust into a cone. Here the dust is emulsified with a small stream of gold-solution tapped from the discharge-column of the same pump, and is then drawn into the suction-line. An automatic float in the cone prevents the introduction of air into the pump-suction. The pump raises the solution

COST OF CYANIDE TREATMENT, ALASKA TREADWELL G. M. CO.

	First period	Second period	Third period
Treatment	Regrinding and amalgamation, followed by agitation in cyanide solution.	Regrinding in cyanide solution, followed by agitation in cyanide solution. (No amalgamation.)	Regrinding in cyanide solution, followed by agitation in cyanide solution. (No amalgamation.)
Period covered	Sept. 16 to Dec. 20, 1910; trial run and first adjustments of the plant.	Feb. 16 to May 15, 1911. (Plant idle Dec. 21, 1910, to Feb. 15, 1911.)	May 16 to July 15, 1911.
Average head value	$54.6174.	$46.9360.	$55.4899.
Average tailing value	$1.7589.	$1.7990.	$1.8892.
Estimated extraction	96.80 per cent.	96.17 per cent.	96.60 per cent.
Tons treated	4906.1.	4620.6.	4080.

First period: Costs Per Ton Treated.

Operation.	Labor (d).	Electric Power, (c) Compressed Air, Steam Heat.	Supplies. Item.	Supplies. Cost.	Sundry Supplies.	Totals.
Grinding	$0.2033	$0.6179	Pebbles...... 8.05 lb.	$0.1182	$0.0449	$0.9843
Amalgamation	0.2324	0.0009	Mercury..... 8.3 oz.	0.1516	0.0066	0.3915
Cyaniding	0.3291	0.1378	Lime.......... 19.78 lb.	0.1978	....	....
			Cyanide...... 2.46 lb.	0.5163	0.0201	1.2011
Filtering	0.3389	0.0791	....	....	0.0233	0.4416
Precipitation	0.1549	0.0645	Filter cloth	0.0162	0.0053	0.3001
			Zinc-dust... 0.79 lb.	0.0592		
Refining (a)	....	....	....	....	....	....
Analyses, assays, accounting (b)	....	....	....	....	0.1780	0.1780
Superintendence	0.1936	....	....	....	....	0.1936
Repairs	....	0.0811	....	....	0.0179	0.0990
Totals	1.4522	0 9816		1.0593	0.2961	3.7892

Second period: Costs Per Ton Treated.

Operation.	Labor (e).	Electric Power, (c) Compressed Air, Steam Heat.	Supplies. Item.	Supplies. Cost.	Sundry Supplies.	Totals.
Grinding	$0.1512	$0.4679	Pebbles...... 22.19 lb.	$0.2686	$0.0061	$0.8939
Amalgamation	....	....		....	....	....
Cyaniding	0.2646	0.1007	Lime.......... 12.77 lb.	0.1226		
	....	....	Cyanide...... 2.86 lb.	0.5665	0.0251	1.0795
Filtering	0.2646	0.0729	Press-cloth.	0.0109	0.0053	0.3537
Precipitation	0.1843	0.0567	Zinc-dust... 2.16 lb.	0.1635	....	
			Filter-cloth	0.0204	....	
			Lead acetate.......... 0.54 lb.	0.0511	0.0134	0.4894
Refining (a)	0.2268	0.0436	Retorts......	0.0619		
			Sulphuric acid..........	0.0617	0.0252	0.4192
Analyses, assays, accounting (b)	0.1269	....	....	....	0.1687	0.2956
Superintendence	0.1890	....	....	....	....	0.1890
Repairs	0.1418	....	....	....	0.0069	0.1487
Totals	1 5492	0.7418		1.3272	0.2507	3.8689

Third period: Costs Per Ton Treated.

Operation.	Labor (e).	Electric Power, (c) Compressed Air, Steam Heat.	Supplies. Item.	Supplies. Cost.	Sundry Supplies.	Totals.
Grinding	$0.1196	$0.1279	Pebbles...... 13.55 lb	$0.1776	$0.0059	$0.4310
Amalgamation	....	....		....	....	....
Cyaniding	0.2093	0.0396	Lime.... 7.06 lb.	0.0649		
	....	....	Cyanide... 2.49 lb.	0.5330	0.0018	0.8486
Filtering	0.2093	0.0316	Filter-cloth	0.0038	0.0059	0.2536
Precipitation	0.1458	0.0234	Filter-cloth	0.0103		
			Zinc-dust... 2.09 lb.	0.1700		
			Lead acetate.......... 0.15 lb.	0.0142	0.0209	0.3846
Refining (a)	0.2243	0.0243	Fluxes.......	0.0193		
			Sulphuric acid..........	0.0656	0.0501	0.3886
Analyses, assays, accounting (b)	0.0748	....	Chemicals..	0.0142	0.0476	0.1366
Superintendence	0.1495	....	....	....	....	0.1495
Repairs	0.1121	....	....	....	0.0274	0.1395
Totals	1.2447	0.2498		1.0729	0.1596	2.7270

(a) During the first period the refinery had not been completed and the precipitate was shipped to a smelter, cost of which is not given.
(b) Assays are made at the central assay office of the combined companies at Treadwell, and charged against the cyanide plant at a proportionate rate per assay.
(c) Electricity, compressed air, and steam are purchased from central power plants, at cost of generation.
(d) Labor, first period, 20 men at $3.75 average wage.
(e) Labor, second and third periods, 22 men at $3.784 average wage. (18 men in mill, 4 men in refinery.)

with the zinc dust to the upper part of the building and forces it through two 36-in. triangular, 16-frame Merrill presses. An average of 145 tons of solution is precipitated daily, with a consumption of 1/8 lb. of zinc dust per ton of solution, equivalent to 0.86 lb. of zinc dust per ton of concentrate. The average strength of solution before precipitation is 1.25 lb (0.0625%) of cyanide; 1 lb. (0.65%) of lime, and $9.50 (9.2 dwt.) gold. The barren or precipitated solutions are kept at 10 cents (2.3 grains), or less, gold per ton, and are used for wash-solution or returned to the Pachuca tanks, as desired.

The Merrill presses are opened when filled or when the pressure exceeds 25 lb. per square inch. Forcing the solution through at higher pressures caused a mechanical loss of precipitate through the canvas. The precipitate is dropped from the press-frames into steel pans and lowered by an electric elevator to the floor below, and thence conveyed by trucks through a concrete passage into the refining-room.

On account of the work required to look after and collect the amalgam, as well as the greater danger of amalgam loss from the pipe-lines and launders, the plates were removed after the first three months' run, and the whole product is now being cyanided directly without amalgamation. In order to handle the larger amount of solution made necessary when grinding in cyanide solution, two 1800-ton steel tanks have been erected, one above and one below the plant. All the precipitated or barren solution flows by gravity from the precipitation-presses to the lower tank. This solution, having an average value of $0.08 in gold, 1.14 lb. of cyanide, and 1.70 lb. of lime per ton, is pumped to the second of these tanks, which is situated 25 ft. above the mill-bins, and acts as the mill-reservoir. Thus at no time is there any cyanide solution run to waste, the solution discharged as moisture in the tailing, plus that absorbed or evaporated in the mill, compensating for that received as moisture in the concentrate delivered to the bins. All the solution used in grinding and classifying is drawn directly from the mill-reservoir. The overflow of fine pulp from the back of the Dorr classifier flows at once to the Callow tanks in the lower building, the spigot-product of which empties into one of the 12 Pachuca tanks for treatment.

The specific gravity of the pulp as it enters the Pachuca tanks is 1.5, or a ratio of 1 of concentrate to 1.18 of solution. The charge is agitated for 8 hr., the necessary cyanide and lime being added to bring the cyanide content of the solution to 1.5 lb. (0.075%) and the lime content to 2 lb. (0.1%) per ton. After agitation and settlement, the clear solution is decanted to the gold-tank through the clarifying-press described later, and a fresh charge of barren solution, the same as that used in the grinding, is drawn from the mill-reservoir, brought to the same strength as the previous treatment, and the charge agitated for 4 hours. This is then settled and the solution decanted. Both solutions decanted from the agitators, together with the overflow from the Callow settling-tanks pre-

viously mentioned, are drawn by gravity through the clarifying-press before emptying into the gold-tanks. The settled pulp in the bottom of the Pachuca tanks, having a specific gravity of 2, is then agitated by means of the spider and pumped to the pulp-storage tank, from which it is drawn to the Kelly presses for filter-treatment.

This method of operation, depending upon the one barren solution for all purposes, keeps the gold-content of the solution to the lowest possible value, which, although contrary to the usual practice, is the object sought in this mill. The solution overflowing from the Callow settling-tanks (containing gold, $10; cyanide, 1 lb.; and lime, 2 lb. per ton) flows by gravity through a special clarifying-press built in the Treadwell shops, the same as receives the decanted solution. This press is of the ordinary plate-and-frame type, yet with a series of ports or channels so arranged as to allow of discharging or sluicing-out a cake without the necessity of opening the press. This sluicing-out press consists of 20 square frames, each 3 in. thick, with the corresponding plates 1 in. thick. The upper channels and two side channels extending through the press have small holes opening into the frame side of the leaf. The upper small channel allows the introduction of compressed air behind the leaves. The lower triangular channel connects with a 6-in. sluicing-out pipe. The press, with connections, is shown in the illustration with one of the plates standing to the left.

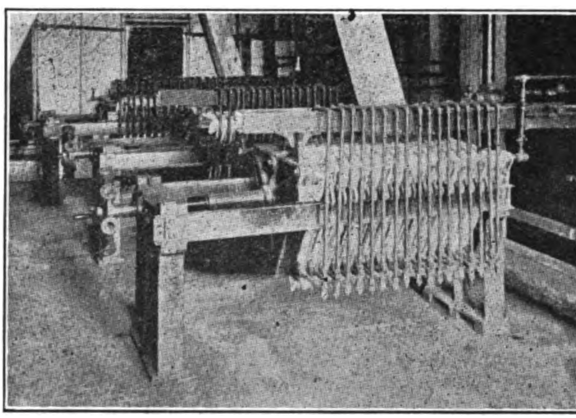
To discharge a cake, water is introduced at the back end of the press through the large triangular opening on the bottom, and flows through the under side to the discharge end, where it empties into the launder leading to the tailing-dam. With this passageway clear, compressed air is introduced through the portholes on the plate side of the leaf. The plate corrugations being depressed 0.5 in. leaves a concave surface, in which the cake forms. The air now being introduced behind the leaves by a series of separate knocks or bumps causes the cakes to drop off into the sluicing-out channel, where they are carried away by the stream of water. For the final washing of the leaves, water is introduced through the three upper channels, and, passing through the tapered holes, is sprayed on the two filter-cloths, which bag together by reason of the compressed air introduced from the plate side.

The method of feeding the zinc dust has been changed somewhat from that originally installed. The reasons for these changes were to create a more even feed of zinc, to do away with the air previously used in the emulsion-cone, and not only to break up any lumps, but to brighten the zinc and grind it even finer. To do this, the drive from the zinc-belt was taken from the Aldrich pump to a small counter-shaft, which was, in turn, belted to a worm-gear for the drive of the zinc-belt, the belt discharging its zinc directly into a small tube-mill 6 ft. long, made from 10-in. pipe, the cast-iron caps of which were turned to run in rollers. This tube is filled with rods of cast zinc 2 inches in diameter. These rods not

only grind the zinc to a more uniform product, but may themselves aid precitipation to a slight extent.

Considerable annoyance is occasioned by the clogging of the cloths in the Merrill gold-presses and by the accumulation of precipitate in the entire line from the zinc-feeder and pump to the presses. Filter-cloths of several kinds—heavy duck at 31c. per yard, various grades of drilling at from 9 to 15c., and muslin sheeting at 7c.—have been tried. The lightest and cheapest muslin is now in use, with results no worse than obtained with the more expensive grades.

From the moment of contact of the zinc dust with the gold-solution, trouble is caused by the slime or precipitate incrusting everything touched. The interior of the pipes gradually becomes smaller in area, even though the solution is driven through at a constantly increasing velocity. After three months' use a 6-in. pipe of 28 sq. in. area was so filled with caked precipitate that only



MERRILL PRECIPITATION PRESSES

a triangular opening of 4 sq. in. remained. From 80 ft. of this pipe, $25,898 was recovered. Being desirous of operating the Merrill presses more or less intermittently without the necessity of each time closing the cocks to retain the solution, which if allowed to drain not only oxidizes the zinc, but causes the precipitate when the pressure is removed to settle in a mass at the bottom of the press-frames, consequently not allowing the greatest amount of solution to pass through the unoxidized zinc, the discharge-cocks were removed from the plates, and open pipes discharging into a launder on top of the presses were substituted, as shown. The result of the several changes is a more uniformly low tailing solution, with the consumption of less zinc, while the gold value of the precipitate has been raised from $15 to $25 per pound; hence a corresponding lowering of refining charges.

The refinery, adjoining the mill, is 30 by 76 ft. in area; constructed of reinforced concrete with steel-truss roof covered with

corrugated iron. The precipitate entering the refinery is crushed through 0.5-in. screen, made up into lots of from 1000 to 1200 lb., weighed, sampled, and charged into one of two redwood tanks, 8 ft. in diameter and 9 ft. deep, conical bottomed, and lined with sheet lead. The tanks are built on the plan of a Pachuca tank, with a central column of wood fitted with lead pipes carrying steam and compressed air for heating and agitating the solutions.

In these tanks the precipitate is treated with acid to dissolve out the zinc, lime, etc. About 1 lb. of 66° sulphuric acid is required per pound of precipitate, and is added in the following manner: About 2 tons of water is introduced into the tank, steam turned on, and the water brought to the boiling-point. Air is turned on the central air-lift, and the acid-valve opened. The acid flows in by gravity, while the precipitate is shoveled in at the rate of 2 lb. of precipitate to each pound of acid. When all the precipitate and from 50 to 60% of the acid have been added, the acid valve is closed and the charge agitated until the acid is entirely neutralized, which generally occurs within 30 min. The tank is then filled with water, and the charge allowed to settle for about 2 hr., after which the clear solution is siphoned off into a filter-tank. The latter is 8 ft. in diameter and 4 ft. deep, having a false bottom of 1-in. strips, placed 12 in. from the bottom of the tank and 1½ in. apart. The strips are covered with heavy iron screen, 1-in. mesh, on which is a bed of burlap 1 in. thick, one thickness of mill blanket, one thickness of light canvas, and a bed 1 in. thick of quartz sand screened between 20 and 80-mesh. The sand is divided into sections of 8 by 10 in. by a light wooden frame, covered by a single thickness of drilling, the latter forming the working-surface of the filter. The solutions filter freely through this medium, the clear filtrate being run into one of three storage-tanks, where it is held until a sample has been assayed, and then run to waste through a series of zinc-boxes. All solutions and wash-waters from the refinery are disposed of in this way.

After decanting the first acid the precipitate in the tank is given two washes of boiling water. Just enough water to enable the charge to be agitated is then added, and the remainder of the acid run in rapidly. This gives a solution containing from 15 to 18% of acid, agitation being continued until the acidity ceases to decrease, which usually leaves about 1% of free acid. The tank is then filled with water, settled and decanted as before. This solution, containing from 50 to 75 lb. of free acid, is at present run to waste.

The charge now receives three or four washes of boiling water followed by washes of about 30°C. temperature, until the wash-water gives no reaction for sulphates with barium chloride, which is generally after 15 washes. After decanting the last wash, the charge is sluiced through a valve in the bottom of the tank on to the filter, which has been thinly covered with silica sand to aid filtration, where the excess water is removed by means of a vacuum-pump. The slime is removed to a large wrought-iron pan, placed upon a 4 by 8-ft. steam-table, inclosed by a sheet-iron

hood. When nearly dry but still damp enough to prevent dusting, the slime is rubbed through a ½-in. screen, weighed and sampled, the weight of the acid-treated product being from 25 to 33% that of the original precipitate. Each lot of precipitate is analyzed before and after the acid treatment, which enables a close calculation to be made of the amounts of fluxes required for the monthly melting.

The percentages of the principal substances contained in an average analysis of the precipitate before and after acid treatment are:

	Before Per cent.	After Per cent.
Au	5.08	17.34
Zn	42.93	5.15
Pb	8.08	20.09
Cu	6.19	14.28
CaO	10.51	1.89
Fe	1.10	0.52
S	1.41	7.62
Insoluble	3.48	22.85

The high percentage of insoluble after treatment is due to the silica added to the lot just before filtering.

At the end of the month the various lots of acid-treated precipitate are united and the various fluxes added. The melting is done in a specially constructed oil-burning furnace. For melting purposes the furnace is fired with a reducing flame. The crucible of hearth used for the melting is 4 by 3½ ft., lined with either magnesite brick or fire-clay, according to the fluxes used. This hearth is placed on a steel car and run under the furnace. Jack-screws, operated by hand-wheels at the four corners of the car, allow of raising the hearth to form the furnace-bottom.

From the fire-box at one end of the furnace the heat is drawn across the top of the charge, being reflected downward by the arch roof and the down-draft to a dust-condensing chamber. The furnace is charged with precipitate hourly, the slag and lead bullion being tapped off intermittently from opposite sides of the hearth. The month's clean-up, amounting to 1450 lb. of acid-treated precipitate, or a total charge, including fluxes, of 2600 lb., is melted on this hearth in 36 hr., and requires the attention of but one man per shift. A typical mixture of fluxes is:

	Pounds.
Acid-treated precipitate	100
Borax glass	22
Sodium carbonate	25
Old slag	80
Iron-turnings	15
Powdered graphite (old retorts)	3

Such a charge will produce about 35 lb. of metal, from 10 to 15 lb. of matte, and from 160 to 180 lb. of slag. From 150 to 300 lb. of high-grade copper matte is produced each month. This matte is roasted and allowed to accumulate until there is sufficient to

make up a charge, when it is mixed with litharge, fluxed, and melted to produce lead bullion, which is the work-lead used for the removal of copper in cupellation. After melting either precipitate or matte, the slag is tapped into conical pots holding about 200 lb., with a tap 4 in. from the bottom, through which the molten core is drawn off. The shells, containing most of the metallic content, are dumped, crushed, and used in fluxing a later charge. The cores, constituting 75% of the total slag, are sampled, sacked, and stored for shipment to the smelter.

The cupellation is done on a limestone test the same size as the melting-hearth, it being run under the furnace on the car previously described. For cupellation the furnace is fired with an oxidizing flame, while free air is introduced over the test by means of a connection from a compressed-air main through a needle-valve discharging into the open end of a 4-in. pipe. This produces low-pressure air, which is introduced into the furnace on the opposite side from which the molten litharge is tapped off. The fine bullion resulting from this cupellation is drawn off and remelted in Faber du Faur tilting-furnaces into bars of 1000 oz. each. The average fineness of the cupelled gold is 880. The retorts of the Faber du Faur furnaces are supported on two 1½-in. pipes built into the furnace, through which cold water is kept circulating. These pipes have proved very satisfactory.

In conclusion, the cyanide-plant has now been in operation one year, using the machines and equipment originally installed, with the exception of the abandoned amalgamation plates, the substitution of larger tube-mill motors, and the addition of the 'Treadwell' clarifying-press, with results summarized in the table. During the month ended August 15, and not included in cost-sheet, 2010 tons were treated, at a cost of $2.8764 per ton, and an estimated extraction of 97.025%, as compared with the experimental estimates of $3.25 per ton and 96% extraction.

TREATMENT OF CONCENTRATE AT THE GOLDFIELD CONSOLIDATED MILL

By J. W. Hutchinson

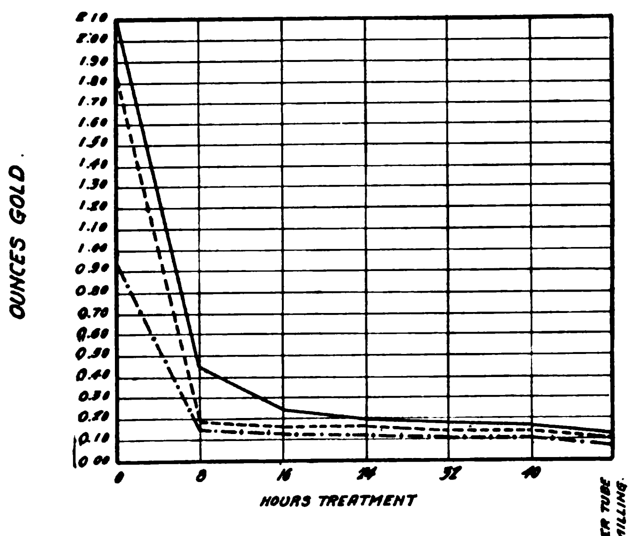
(January 25; February 1, 1913)

Since this article is intended to supplement the series* which appeared in the *Mining and Scientific Press* during May and June of 1911, those interested are referred thereto for a description of the former process for treating concentrate.

It will be necessary to say here only that the acid wash, bromo-cyanide, and peroxide that were formerly applied to the raw concentrate have been discontinued. The concentrate, previous to roasting, is subjected to a preliminary treatment with alkaline cyanide for the purpose of reducing the value of the material to be roasted in order to minimize the dust loss. For instance, the loss

*Reprinted in this volume under 'Description of Notable Mills.'

in roasting has been determined to be approximately one-half of one per cent of the value of the material roasted. When the roasting plant was put in operation the raw concentrate entering the plant assayed 10 oz. in gold. The loss on concentrate of this grade would have been $1 per ton. The preliminary treatment given before roasting removed 85% of the value, leaving a product to be roasted valued at $30, and the loss in roasting this material was 15c. per ton. Further, it was demonstrated conclusively in the laboratory that neither a complete raw treatment nor treatment of the rich concentrate after roasting would yield so complete recovery as the combination of the processes, and that this combination treatment could be given as economically as either of the single treatments. For this reason it was decided to dispense with all the more expensive details of the raw process and to roast the tailing from this modified raw treatment.



The raw concentrate recovered from the 78 Deister concentrators (the 16 secondary tables are not now used), amounting to 6% of the weight and containing 67% of the value of the ore, is collected in flat-bottomed tanks, equipped with the adjustable square shaft agitator, which was fully described in the former articles; it is here neutralized with lime and by means of a centrifugal pump elevated to three Pachuca agitators, in which it is agitated during 8-hour periods in a two-pound solution of cyanide. Decantation at the end of the period is still practiced and the charge is re-agitated with a freshly precipitated solution. Five periods of 8 hours each, followed by decantation, are sufficient to remove from 80 to 85% of the value of the concentrate. It is the intention to send to the roaster a product valued at $25 to $30, and

this treatment is varied with the grade of the ore so as to accomplish this result. The pulp from the Pachucas, when dissolution is completed, is delivered to a storage tank from which it is pumped to the Kelly filter-press for filtration and drying. The latter is accomplished with air and the moisture is reduced to 12%. The consumption of cyanide during the raw treatment is 2½ lb. per ton of concentrate, and lead acetate is used in the proportion of 1 lb. per ton.

By means of a hoist and scraper, approximately 20 tons per day of the accumulation of concentrate on the dump is conveyed to the Kelly press bin and mixed with the concentrate filtered. A 14-in. conveyor, set at an angle of 17°, receives the concentrate from this bin, passing it over a Blake-Dennison automatic weighing machine en route to the bins in the roaster plant. The concentrate from this conveyor is distributed by means of a swinging bucket elevator to two bins having 45° sloping bottom sand 1620-cu. ft. capacity, from which it is fed by means of two 12-in. screw-conveyors, making three-quarters of a revolution per minute, to two slow-moving belts. These belts discharge the concentrate through the arches of the furnace between the first two rabbles. The original feeding arrangement was a 12-in. screw-conveyor, 23 ft. long from the ore-bin to the furnace. The strain caused by the packing of the concentrate made this machine impractical. The screw was cut down to 10 ft. in length and used for feeding only, and the concentrate is now conveyed from this feed-screw to the furnace by the belt above mentioned.

Roasting is accomplished in two Edwards (54 spindle) duplex furnaces, 112 ft. long by 13 ft. inside, with an effective hearth area of 1456 sq. ft. each, and each furnace is capable of roasting 40 tons per day of concentrate, of which the following is a typical analysis, the average sulphur content being 18.76 per cent:

Mesh.	Weight, per cent.	Sulphur, per cent.	Total S, per cent.
100	9.5	5.34	2.70
150	8.5	11.74	5.27
200	22.5	18.59	22.28
—200	59.0	22.37	70.30

The hearth area required per ton of concentrate roasted per day is 36.40 sq. ft., equivalent to 55 lb. of concentrate per square foot per day. The slope of the hearth is ¼ in. per foot. The space between the furnace side-walls was filled with waste from excavations to within one foot of the hearth-line. This material is decomposed surface rock full of clay. It was wetted down and tamped thoroughly and covered with one foot of screened sand to form the hearth, in order to reduce the breakage of revolving parts. Since all details of construction can be obtained from the accompanying figures, there is no necessity for repetition. There are two rows of 27 rabbles; 25 of these revolve at 2.25 r.p.m. and the last two on the finish at 4.5. This speed was decided on after testing the fur-

naces at speeds varying from 1.6 r.p.m. to the speed now used. Lower speeds did not reduce the sulphur in the discharged product, decreased the capacity, and in no way affected the dusting. In addition to being a most satisfactory furnace to operate, the Edwards has the distinct advantage of producing a minimum of dust. The fear of serious loss from this feature of roasting caused the delay in adopting the two-stage treatment. That is was groundless has been proved by operation. Only 1½% of the material roasted passes out of the furnace as dust, and only ½% is lost. When the accompanying analyses quoted and that below are considered, the performance seems remarkable:

Mineral.	Percentage.
SiO_2	51.60
Fe	19.90
S	18.93
Al_2O_3	2.00
CaO	0.20
MgO	0.10
Sb	0.08
Te and Se	0.46
Cu	0.50

One man per shift operates the entire plant, including feeding, firing, oiling, and attending cooler and elevator. The bins are filled on the day shift and have sufficient capacity to run for 24 hours. Each furnace receives power from a 10-hp. motor, and including the feed-screw and belt, which are driven from the furnace shaft, requires 4½ hp.

At the start all the fire-boxes were used for several weeks. The first to be discontinued were the middle boxes. For several months thereafter fuel was burned in the front boxes and on the finishing hearth. As stated above, the moisture in the material fed the furnace averages 12%. Approximately 6 sq. ft. of hearth area per day ton is required to remove this moisture, and the concentrate is not thoroughly dried until the fourth rabble is reached.

An attempt was made to discontinue the fire in the front box, with the result that the moisture traveled farther down the furnace and a decided decrease in capacity was caused. It was then decided to dispense with the fires on the finishing hearth and to burn all the oil required in the front boxes. This is the present practice and has resulted in reducing the fuel consumption approximately 45%. Nine gallons of crude oil is required per ton of concentrate, equivalent to 3.3% of the weight. The sulphur begins to oxidize at the seventh rabble and is burning freely at the tenth. Doubtless because of the extremely fine state of division of the sulphide, after once becoming ignited, roasting is carried to satisfactory completion without additional fire at the finishing end. This, of course, is contrary to the usual practice when roasting previous to cyanidation, since it is customary to raise the temperature at this point, but it has been demonstrated conclusively here to be the most economical practice for this material. Were roasting ahead of chlori-

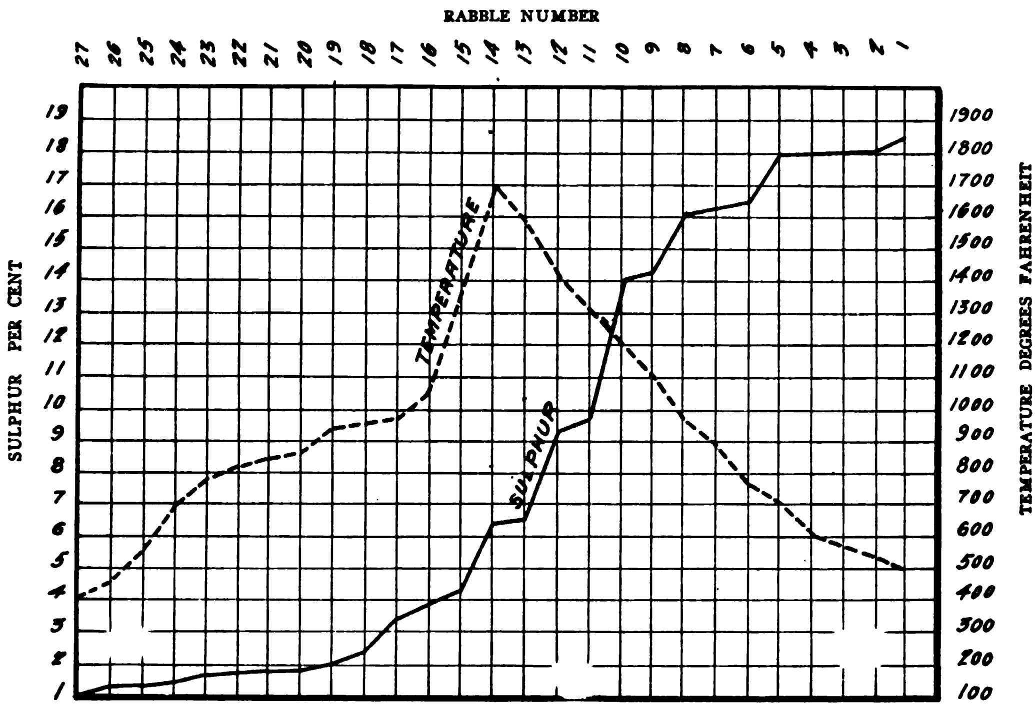
nation being done, the scheme would not be feasible, since a dead roast would be required and the soluble sulphur would be abjectionable. However, at this plant the treatment required by the roasted material prior to cyanidation, removes the sulphate and obviates the necessity of a complete roast. It is no doubt true that the fineness of the pyrite makes this practice possible and that coarser concentrate would require different treatment in the roaster. E. D. Peters in his 'Principles of Copper Smelting' makes the following observation:

> Theoretically, the smaller each particle of sulphide, the more rapid and thorough will be its oxidation. It takes an appreciable time for the oxidation process to penetrate into the centre of even a very small particle of the sulphide mineral. It begins its work upon the surface, and the more surface it is allowed to operate upon, the more oxidation there will be in a given space of time.* * * Consequently, if there were no disadvantages arising from the employment of very finely pulverized ore, it is evident that these would be no limit to the fineness at which it would be advantageous to have our ore for reasting; so that, if it were as fine as soot or rouge powder, it would roast instantaneously, * * * but in practice there are difficulties which more than offset the advantages arising from the rapid and thorough oxidation of this very fine ore. Among the more obvious of these difficulties are: the great expense of pulverizing ore to this fineness; the enormous production of flue-dust in the roasting furnaces from the fine particles carried away by the draft; the tendency of the sulphide particles to melt from the heat arising from instantaneous oxidation; the fact that such excessively fine ore lies solid in the roasting furnace, offering no interstices for the penetration of air into the deeper layers.

As can be seen from the screen analysis, the product fed the furnaces has been pulverized so that 80% passes a 150-mesh screen and this 80% contains 92½% of the total sulphur. Obviously, the problem here has been to avoid excessive loss from dusting and the melting of particles. In the matter of admitting air to the hearth, the practice here varies from the usual in that all side ports are kept closed, except six near the finish. In this way, most of the air entering the furnaces is preheated by passing over the practically roasted ore before it reaches that part of the furnace where heat is required, and thus saves fuel by not excessively reducing the temperature at this point. With a given degree of comminution the factors governing the oxidation of pyrite are, time, temperature, and oxygen. The first is inversely proportional to the last two. For instance: Oxidation at the surface of the earth is accomplished in long periods of time at low temperature and an abundance of oxygen; the same chemical result can be obtained in an assay muffle in 60 minutes with high temperature, while in practice the result is obtained in a number of hours. Since the combustion of sulphur produces dead atmosphere, and since combustion ceases entirely when the amount of SO_2 in the surrounding air reaches 12%, it follows that oxidation proceeds more rapidly in pure air than in air diluted with reducing gases. However, since a large excess of air or draft increases fuel consumption and stack losses, the economic limit of any furnace is reached when the decrease in cost of labor and power resulting from increased capacity is more than offset by the increased cost of fuel and increased stack

losses. This point has been demonstrated here by repeated tests. There is no difficulty in increasing the capacity of the furnaces by burning more fuel and increasing the draft, but there is no resultant economy from such increase on this extremely fine material. Since the element of time is not only essential, but an economical factor in the commercial oxidation of pyrite, it seems that where the tonnage to be treated is sufficient to justify the increased capital outlay, greater economy can be effected by allowing greater hearth area per day-ton of concentrate roasted. The material roasted here

CHART SHOWING SULPHUR ELIMINATION AND TEMPERATURES AT THE RABBLES



is so out of the ordinary and requires such unusual treatment, it has been thought advisable to go quite into detail in order to explain the theory on which the practice is based.

The accompanying chart of temperatures and sulphur elimination will show the progress of the concentrate through the furnaces. All temperatures were taken with a Brown electric pyrometer, using a platinum-rhodium couple, and for the sake of uniformity the readings were taken one foot above the hearth-line. The concentrate loses 17% of its weight in roasting.

By means of iron goose-neck flues, the gases from the roasters at a temperature of 450°F. are delivered to a concrete dust-flue

264 ft. long, having a cross-section of 50 sq. ft. From this flue, 20,700 cu. ft. of gases per minute escape through a vertical steel stack, 100 ft. high and 54 in. diam., having a temperature at the base of the stack of 325°F. The velocity of the gases in the dust-flue has been determined with a Hiram anemometer to be 7½ ft. per second. The escaping gases are white, and no dust is visible to the eye. Measured quantities filtered through woolen bags indicate a stock loss of less than ½%. This figure is taken as a total loss in handling, roasting, conveying, etc., and added to the tailing loss. One per cent of the material roasted is collected in the dust-flue and has a value of 20% more than the material roasted. Apparently, all the dust is made in the feed-end of the furnace, since this dust contains 15% sulphur. Approximately 80% of the dust recovered settles in the first 150 ft. of the flue. The flue is built on the slope of a hill and at the top it is 79 ft. above the hearth. As can be seen from the drawing, a drag-chain conveys this dust down the flue, from which it is discharged through wrought-iron gates into a sump. Water is added and the resultant pulp elevated back to the Kelly press storage tank for filtration and mixing with the regular feed. The loss and inconvenience through handling this fine material dry is thus avoided.

Analysis of Roasted Product

Mesh	Per cent.
+ 100	12.5
+ 150	15.5
+ 200	25.5
— 200	46.0
Composition.	
SiO_2	54.60
Fe_2O_3	32.20
S (as sulphide)	0.15
S (as sulphide)	0.75
Al_2O_3	3.00
CaO	0.20
MgO	0.13
Cu	0.60
Te and Se	0.19
Sb	0.07

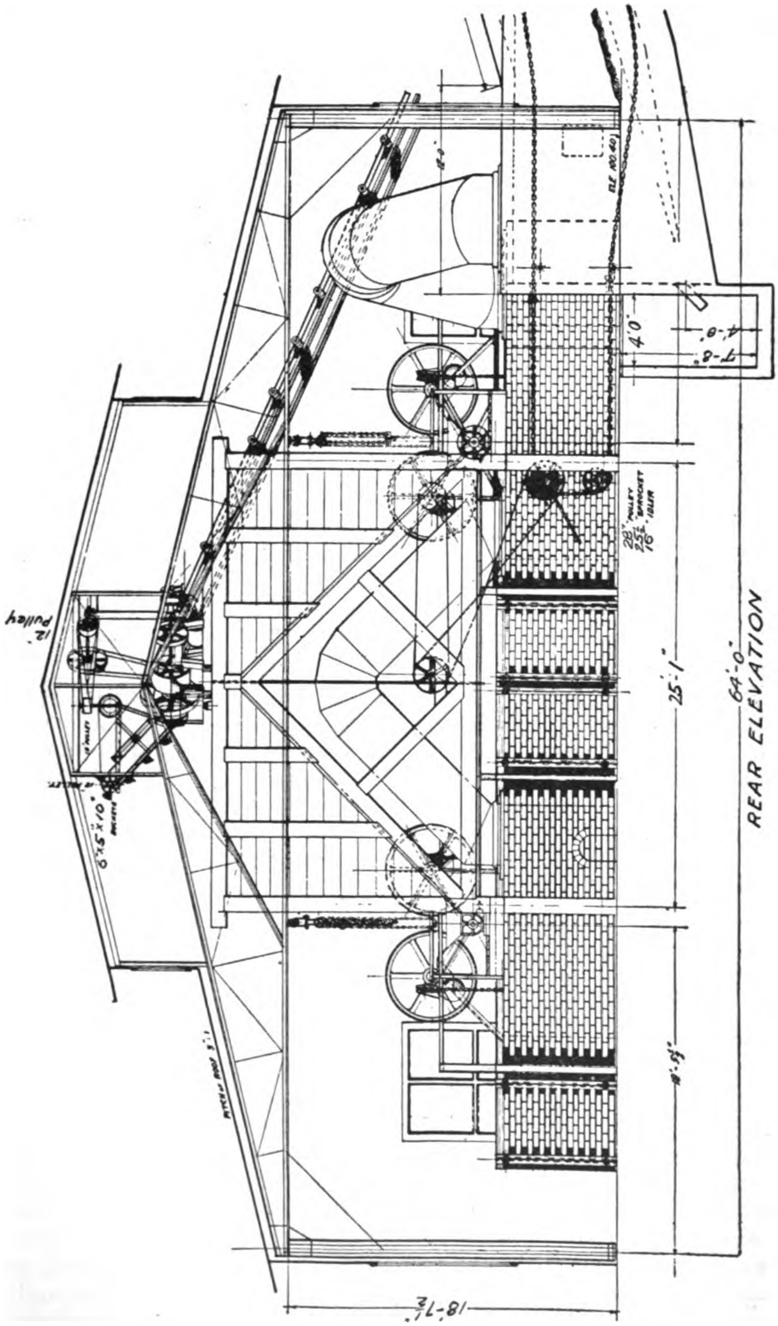
In line with the last rabble on each furnace there is a cast-iron discharge chute on one side of the furnace only, set at an angle of 45°, through which the roasted material is delivered to a drag-chain conveyor. This conveyor runs at a speed of 45 ft. per minute and elevates the material up an incline of 15° to the feed spout of a Baker cooler. As can be seen from the drawing, the original device for conveying the hot concentrate was a water-cooled screw-conveyor. This was decidedly unsatisfactory and was discarded in favor of the drag-chain. This chain does good work, requires no repairs, and is very satisfactory. The drag-chain, sprockets, and all movable parts are housed in a brick conveyor-way built up 2 ft.

above the floor line and covered with ¼-in. plate, and no dust escapes into the building.

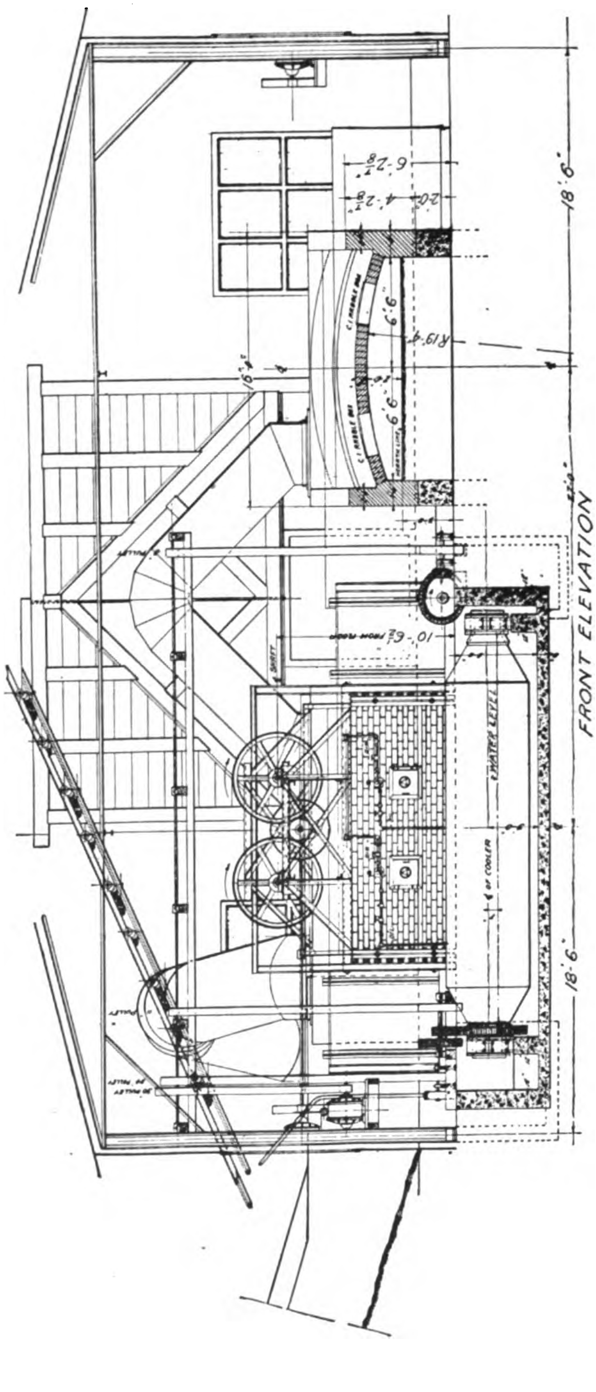
The Baker cooler is sheet iron cylinder, 5 ft. diam. and 22 ft. long, carried at each end on a hollow trunnion bearing, through which the ore is fed and discharged. It revolves with about 40° submergence in a water-tight concrete sump, to which the cooling water is added. Cooling is accomplished through the evaporation of water on the surface of the shell. The cooled ore is discharged dry at about 100°F. into a small concrete sump, over which there is a tight cover, where water is added to wash it to the boot of a 14-in. belt and bucket elevator with 12-ft. centres. Much difficulty was experienced in elevating the material after wetting, due to the soluble copper compounds in the roasted product. Cast-iron centrifugal pumps would not last for this reason. Phosphor bronze pumps were too soft to resist the abrasive action of the pulp. A temporary belt and bucket elevator with steel parts lasted a couple of weeks. Finally, by using brass pulleys, shafting, cups, bolts, and a canvas belt, the elevators have been made fairly satisfactory. The small elevator in the roaster building delivers the pulp to the boot of a second elevator of similar construction placed in the treatment plant, which delivers it to three 20 by 12-ft. combined collecting and agitating-tanks, fitted with the adjustable square shaft agitator. The sole plates, holding bolts for these agitators, are made of brass, and the shaft is protected from the corrosive action of the washes by means of a lead covering. The tank connections in these tanks are brass.

The roasted concentrate is delivered to one tank for 24 hours. The collected charge is then settled and decanted to a consistence of 1 to 1 and sulphuric acid added in the proportion of 20 lb. per ton of concentrate. Agitation with the sulphuric acid is continued for eight hours. Water is then added to fill the tank and the charge allowed to settle. When clear, the wash is decanted and the tank re-filled with fresh water. Four water washes are given, equivalent to eight tons of wash water per ton of concentrate. All washes are passed through two redwood tanks filled with excelsior for clarifying ,and overflow from these tanks to six redwood tanks, 10 ft. diam. and 5 ft. high, arranged in series for recovering the copper. These tanks are kept filled with cyanide tins and all kinds of scrap from the mill. The average copper content of the washes is 0.4 lb. per ton, and 70% is recovered.

The thoroughly washed charge is neutralized with lime, and by means of centrifugal pumps elevated to one of four Pachuca agitators, 14 ft. diam. by 25 ft. 6 in. high. Experimental work showed that only 10% of the gold of the roasted concentrate could be recovered by amalgamation, but that no increase in final-extraction could be obtained by this step, and for this reason amalgamation was not deemed necessary. The only explanation known for this peculiar action of the gold is that the antimony, bismuth, tellurium, and selenium (though present in minute quantities only) with which the gold is so intimately associated, are not completely



CROSS-SECTION OF FURNACE SHOWING DISCHARGE END



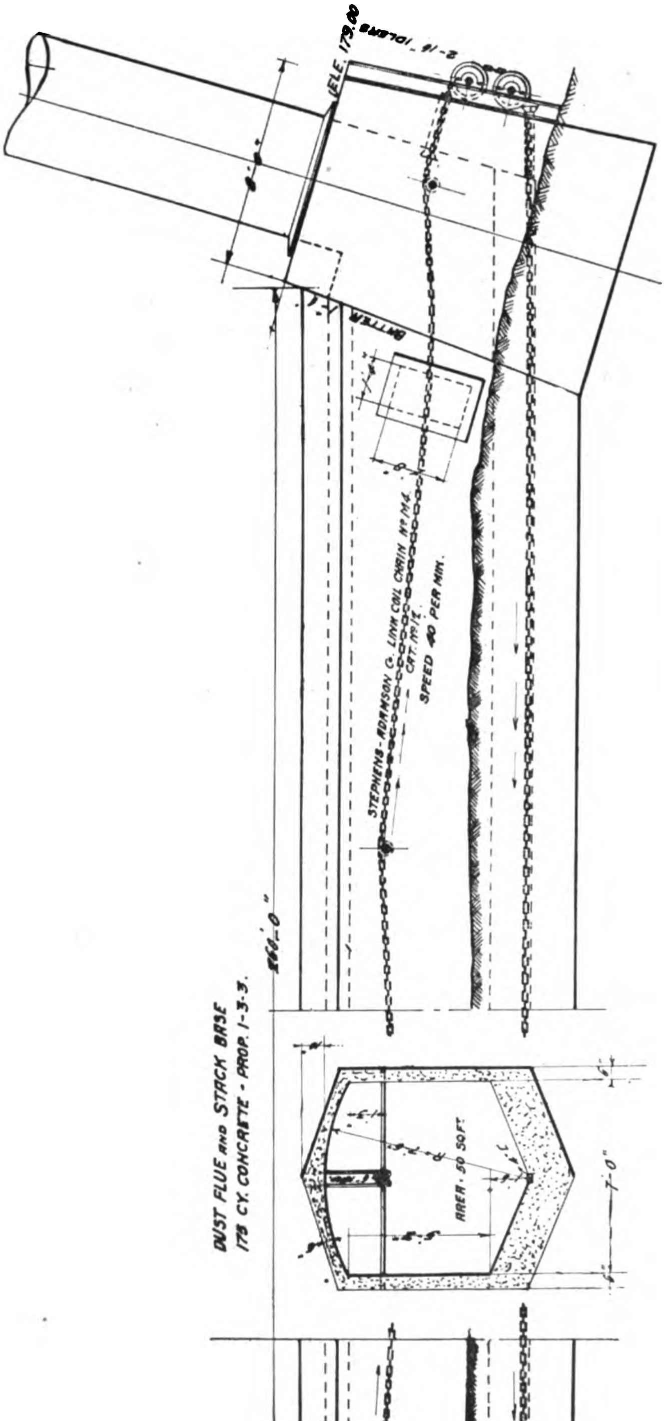
CROSS-SECTION OF FURNACE SHOWING FIRE-BOX

volatilized in the furnace, and form a film of their oxides on the surface of the gold particles. The same explanation will account for the failure of the rich roasted concentrate to yield so high a percentage of its gold to cyanidation, as can be recovered from the two-stage treatment. The following comparison is interesting:

	Au, oz.	
Assay before roasting	12.25	
Assay after roasting	15.35	
Assay after 64 hr. treatment	0.66	
Percentage recovered		95.7
Assay of raw concentrate	11.60	
Assay after 48 hr. treatment	0.96	
Percentage recovered		91.7
Assay before roasting	0.96	
Assay after roasting	1.17	
Assay after 32 hr. treatment	0.107	92.6
Percentage of original gold recovered by raw treatment		91.7
Percentage of original gold recovered after roasting		7.6
Total percentage recovered		99.3

It is a fact proved by repeated experiments and by working tests that no more gold can be amalgamated from a 10-oz. roasted concentrate than can be amalgamated from the same material in its raw state, and that roasting the rich concentrate does not materially expedite the dissolution of the gold. It seems probable that some of the gold is partly encased in the sulphides of the volatile metals, but presents some surface to which the cyanide solution has access. After roasting, it is probable that the gold may be entirely covered with a film of the oxide of such metals, which must be removed before dissolution can proceed. Since these metals do not interact with alkaline cyanide, their removal is doubtless accomplished slowly by the alkali of the solution, and this may account for the prolonged treatment which the roasted material requires. This theory is offered merely as an explanation and the ideas of others would be appreciated.

In the above mentioned Pachuca tanks the roasted charge is agitated for eight hours in a 2-lb. solution of cyanide, containing 1.2 lb. CaO as protective alkali. At the end of eight hours, agitation is discontinued, the charge settled, decanted, and re-agitated with a freshly precipitated solution in the same manner as described above in the treatment of the raw concentrate. Five periods of agitation followed by decantation are given, and a total of three tons of solution per ton of concentrate is decanted. Consumption of chemicals amounts to 4½ lb. cyanide and 2 lb. lead acetate per ton of concentrate. After agitation is completed, the settled charge is delivered to a storage tank 18 ft. diam. by 8 ft. high, fitted with the adjustable square-shaft agitator. Placed centrally in the bottom of this tank is a 4-ft. cone with pipe connections through which the thickened pulp is fed to a 5 by 18-ft. tube-mill. The pulp issuing from the tube-mill is elevated by



LONGITUDINAL AND CROSS-SECTION OF DISCHARGE DEVICE

means of a belt and bucket elevator back to the above mentioned storage tank. This circulation grinding is continued for 16 hours, at the end of which time 95% of the material will pass a 200-mesh screen. From 80c. to $1.25 per ton is removed in this circuit. Re-grinding before agitation with cyanide was the original plan, but had to be discontinued on account of the inability to settle and decant the finely pulverized ore. Since the change of solution increases extraction and since the final tailing is sent to the mill proper for filtration, it was decided to regrind after the greater part of the gold had been removed. After re-grinding, the pulp is delivered by means of a centrifugal pump to the filter storage tank in the mill proper, mixed with the mill pulp, filtered, and sent to waste.

Since precipitation and refining are accomplished in the manner described in the former article, it is not necessary to give the details of these operations here. Extraction and costs are shown below:

Cost per Ton

Labor		$1.02
Power		0.78
Cyanide	$1.37	
Zinc	0.06	
Lime	0.25	
Lead acetate	0.16	
Water	0.70	
Belting	0.02	
Lubrication	0.01	
Borax	0.01	
Litharge	0.02	
Pig lead	0.01	
Pebbles	0.04	
Tube-mill lining	0.00	
Filter cloth	0.03	
Assaying	0.13	
Acid	0.48	
Fuel oil	0.40	
Roaster parts	0.01	
Bromo-cyanide	0.00	
Sodium peroxide	0.00	
General stores	0.18	
Total supplies		3.88
Total costs		$5.68
First agitation		$0.73
Filtering and conveying		0.30
Roasting		0.82
Acid wash		0.77
Tube-milling		0.33
Second agitation		1.61
Assaying		0.15
Precipitation		0.10
Refining		0.08
Disposal of residue		0.03
General expense		0.02
Re-handling dump		0.04
Water		0.70
Total		$5.68*

*This may be divided into: operation, $5.32; repairs, $0.36.

Extraction for Year

	Au, oz.	Per cent.
Value after treatment	1.23	
Value of raw concentrate	6.58	
Recovery		81.3
Value of tailing after roasting and treating	0.097*	
Recovery		92.16
Total recovered from roasted material		17.23
Recovered from both treatments		98.53

*Based on weight before roasting.

Cost of Roaster Plant

Contract for:		
2 Edwards duplex 54-spindle roasting furnaces, 112 by 13 ft. inside erected on side walls furnished by the Goldfield Con. M. & T. Co., which side walls extend to within one foot of hearth line	$33,485.17	
1 Baker cooler, 5 by 22 ft., erected on foundations furnished by the Goldfield Con. M. & T. Co., complete with driving mechanism, etc	3,400.00	
Excavations for foundations	1,897.00	
Concrete in foundations, 218 cu. yd. at $14.25	3,106.50	
Total cost of furnaces and cooler erected		$41,888.67
Dust-flues and stack:		
Dust-flue of concrete, 264 ft. long, 6 ft. high, 7 ft. wide; walls 7 in. thick, bottom 10 in. thick with 24° slope to centre; roof 8 in. thick, arc of 10 ft. circle; concrete reinforced with ¼-in. rods, spaced 1 ft. apart both horizontally and vertically, and with a steel cable laced through these rods; total cost	$ 5,500.00	
Stack of American ingot iron, 100 ft. high by 54 ft. in diam., weight 10,620 lb., invoice $573.60, freight $519.60; total cost	1,093.20	
2 goosenecks, 60-in. diam. and 42-in. diam., weight 4480 lb., invoice, $416.00, freight $248.49; total cost	664.49	
Supplies used in erection	375.20	
Labor	311.48	
Total cost of dust-flues and stack		$ 7,944.37
Steel building:		
65 by 158 ft., containing 10,270 sq. ft., at $7398.23 erected in Goldfield	$ 7,398.23	
Foundations	1,213.87	
Covering of 'asbestos protected metal,' weight 1¾ lb. per sq. ft.	3,538.84	
Labor for laying asbestos covering	700.00	
45 windows in place at $10 each	450.00	
		$13,300.94

Ore-bins:		
2 wooden ore-bins, 1620 cu. ft. cap.:		
Supplies	$ 721.26	
Incidentals	55.93	
Labor(erection	311.70	
Total cost of ore-bins		$ 1,088,89
Transmission, elevating, and conveying machinery:		
Line shafting, etc.	2,069.87	
Labor, erection	1,858.55	
Total cost		$ 3,928.42
Miscellaneous expense:		
Labor	$ 530.50	
Supplies	1,086.81	
Total miscellaneous expense		$ 1,617.31
Electrical work:		
Labor	$ 163,00	
Supplies	527.56	
Total electrical work		$ 690.56
Total roasting plant proper		$70,459.16

Cost of Treatment Plant

Superstructure:		
Wooden building, 44 by 48 ft., including lumber, concrete, and covering	$ 3,109.13	
Labor (erection), excavating, moving dumps, etc	2,447.48	
Total cost of superstructure		$ 5,556.61
Pachuca agitation:		
2 steel Pachuca agitators, 14 by 25½ ft., weight 32,860 lb.:		
Invoice	$ 1,225.00	
Freight	384.46	
Labor (erection)	1,316.65	
Power (erection)	200.00	
Labor (foundations)	477.70	
Supplies (foundations)	283.87	
Other materials	533.61	
Total costs		$ 4,421.29
Redwood collector and agitation tanks:		
2 redwood tanks, 20 ft. diameter by 12 ft., weight 36,000 lb.:		
Invoice	$ 402.00	
Freight	265.00	
Labor (erection)	505.30	
Stirrers and other suppiles	1,084.11	
Labor (foundations)	496.40	
Supplies (foundations)	197.27	
Total		$ 2,950.08

Tube-bills:		
5 by 18-ft. Gates tube-mill, weight 25,287 lb.:		
Invoice (including chain drive)...	$ 1,910.00	
Freight	650.73	
Labor (erection)	297.55	
Supplies (erection)	114.13	
Labor (foundation)	298.45	
Supplies (foundation)	258.94	
Labor (lining)	111.35	
Supplies (lining)	301.64	
Total		$ 3,942.79
Miscellaneous:		
Labor	$ 530.50	
Supplies	1,086.81	
Total		$ 1,617.31
Total cost of treatment plant....................		$18,488.08
Total cost of roasting plant.....................		70,459.16
Total cost of treatment and roasting plants		$88,947.24

ROASTING

ROASTING AT KALGOORLIE

By 'METALLURGIST'

(July 9, 1910)

*The discovery of telluride ore on Block 45 in 1896, and a little later on in the Lake View Consols mine, set the metallurgist experimenting as to the most suitable method for the treatment. Various schemes were tried, with varying amounts of success, until J. W. Sutherland, metallurgist of the Lake View Consols, tried roasting in the ordinary assay muffle furnace. After such roasting the ore was readily treated by amalgamation and cyanidation, and this induced him to make experiments on a large scale. In 1897 the first roasting furnace on the Golden Mile was erected on the Lake View mine. The furnace was after the style of the old hand-rabbled reverberatory of Victoria, the length being 15 ft. and width 11 ft. As it was necessary to dry-crush the ore before roasting, the Lake View loaned one of the ball-mills at the Associated mine and crushed the ore through a 1/8-mesh screen. The results of roasting larger quantities were equally successful, and it is on these results that the foundation of the present roasting plants is based. It being decided in 1898 by Mr. Callahan, manager of the Lake View Consols, to install a properly equipped roasting plant, instructions were given to Mr. Pratt, the engineer, to proceed to the Eastern States and inspect and report on the roasting plants then in use. While he was making his report the Lake View Consols office was in communication with Europe, and when the whole of the information was available a Brown straight-line furnace was ordered, and in 1899 the two furnaces started roasting on a large scale, and had not the Swansea people supplied a rope in place of chain for the rabbles, the obtaining of a sweet roast would not have been so seriously delayed. It is not intended to go into technicalities, but nearly every one is conversant with the varying results obtained, sometimes necessitating the blasting out of the sands in the leaching vats. While Mr. Pratt was away the Great Boulder Main Reef decided on a roasting plant, and sent Mr. Marriner, the metallurgist, to the Eastern States to inspect and report, and it was on his report that the Mt. Morgan shaft-furnace was installed on the Main Reef in 1899. In connection with this furnace, one of the newspapers went so far as to say that the Great Boulder Main Reef was the first mine here to pay a dividend on sulphide ore, and the extractions were given out at 96% of the gold content, although later investigations qualified these reports. While these two mines were roasting and completing the roasting plants, the Associated and Kalgurli mines were erecting the Ropp furnaces, the Perseverance, the Holtoff-Wethey, and the South Kalgurli the Brown straight-line. It was during 1900 that these plants were put into operation. The Great Boulder had pinned its faith to the Kooneman process, and under Mr. Kooneman's superintendence was erecting a gas-fired vertical-

*Abstract from *'The Gold Mines of Western Australia.'*

column furnace somewhat similar to the Stetefeldt. This was designed for roasting the ore in particles as large as nuts and to do away with the fine grinding which was necessary with all the other plants. Unfortunately, this plant soon proved a failure, and in 1900 the management chose Mr. Lilburne, the metallurgist, to take a sample of the Great Boulder sulphide ore to Ballarat to be roasted in the Edwards pyrite works. The trials were successful, and on his return the erection of a set of Edwards furnaces was undertaken. A feature of these furnaces was to be the gas-firing and a gas-plant was installed, but the economy of gas fuel was not the success anticipated, and after a lot of hard work the officials had reluctantly to revert to wood for fuel.

In 1901, with all the different types of furnaces roasting, the metallurgists were busily engaged in obtaining best results from the furnaces, and in comparing results with the adjacent mines. The mines were rapidly developing large sulphide orebodies, and an increase of roasting plant was becoming urgently necessary. The managements were anxious that the additional furnaces should be of the best of the different types, so naturally the roaster and roasting was the most prominent metallurgical topic. The Kalgurli mine was the first to admit that the Ropp was not suitable for the work, and in making additions to the plant discarded this type entirely, and erected in its place the Edwards furnace, with a single line of rabbles, and with raising and lowering gear to alter the furnace hearth to any level which was thought necessary. Nine of these furnaces started work early in 1901. At about this time the Ivanhoe, which had been running a small tower furnace with a capacity of three tons per day, since February 1900, and the Brownhill, where the stampmill and wet-crushing with concentration was in use, decided that the concentrate being produced could be more profitably treated by roasting at the mine than by shipping to the custom works, and placed an order for Edwards furnaces. The results obtained after this change proved eminently satisfactory, the furnaces at the Ivanhoe being still in use. A fourth furnace was added in 1902 and a fifth in 1906, making a total of five Edwards simplex furnaces in use at the present day. The Great Boulder management at this time was also anxious to increase the monthly tonnage of ore treated, and not being altogether satisfied that the Edwards furnace was the best, instructed G. M. Roberts to investigate and report on the Merton furnace. On his recommendation the company decided that the additional furnaces should be of the Merton type. These furnaces were duly installed and were working in 1903. The Associated company, now being thoroughly convinced that the Ropp furnace could be improved upon, were making inquiries as to what should be installed. All the furnaces were getting fairly good results, and in making the choice the company, helped by G. M. Roberts, who pronounced strongly in favor of the Merton, decided to erect ten of this type. These were complete and running in 1904. The Kalurgli had decided again to increase the monthly tonnage, and the South

Kalgurli to overhaul its mill and make it up-to-date, and the question of the best type of roaster was exercising the minds of F. A. Moss on the Kalgurli and J. Morgan on the South Kalgurli. The Kalgurli decided on installing the Edwards furnace with a bricked-in hearth in place of the tilting furnace. The South Kalgurli replaced the Brown straight-line with the Merton. These alterations and additions were completed in 1905. The Golden Horseshoe, which was using the stampmill and concentrating, had decided to follow the lead of the Ivanhoe and Brownhill and roast the concentrate from the Wilfleys at the mine. The management approved of the Edwards plan of working with two lines of rabbles, in place of the single line as used at the Kalgurli and other mines. After going into the plans with the maker, J. W. Sutherland placed an order and constructed the first duplex furnace for roasting concentrate at the Golden Horseshoe in 1905. The results of this furnace were so excellent that the Perseverance, seeing the improvement that was possible with the double line of rabbles, made drawings for the replacing of the rabble of their Holtoff-Wethey furnaces by the Edwards duplex, and the using of the lower hearth as a conveyor and cooling surface. This alteration was successfully carried out by G. C. Klug in 1905, this being the first duplex furnace roasting sulphide ore on the Golden Mile.

The Associated was then putting in another Merton. The Associated Northern decided in favor of the Merton, while the Oroya-Brownhill replaced its Edwards by a Merton. The furnace practice had now settled down to the two types, with the exception that Mr. Dagger, of the Associated, had designed and erected a two-hearth furnace, practically a combination of the Merton and Edwards, and called it the Associated furnace. Each furnace finds strong support with masses of figures of costs and extraction to bear out the claims of the makers. It looked as if the two types would have equal support for all time, but Mr. Hamilton, of the Great Boulder, wishing to increase the tonnage and having both types of furnace in use, having decided on erecting a duplex Edwards; and Mr. Roberts, of the Associated, the strong supporter of the Mertons, following his example, settled all arguments and pronounced the Edwards duplex as being the most suitable furnace for the Kalgoorlie sulphide ores.

As showing the enormous strides during the last twelve years that have been made in the roasting practice, the small reverberatory on the Lake View, and the up-to-date plants working at the present day may be contrasted. While the industry of Western Australia has reaped the benefit of all the work done by the engineers and metallurgists in this line, it may not be out of place to mention that at Cripple Creek, in America, where telluride ore was known to exist at a much earlier date than the Golden Mile, the method of treatment adopted was the smelting of the rich product only. That field has partly adopted the Kalgoorlie method of treatment; and not only can the mines be worked at a greater profit, but dumps formerly valueless are being treated at large profit by wet crushing and concentration.

At the present time there are working on the Golden Mile plants treating a total for the district of 73,000 tons of crude ore each month, and 7300 tons concentrate by roasting, a record for any part of the world, and, considering the high cost of material, the costs and extraction have not yet been equalled by any other mining center. The detailed equipment is as follows:*

The Great Boulder G. M.: 12 Edwards simplex; 8 Mertons; 3 Edwards duplex, roasting 18,000 tons sulphide ore per month.

Great Boulder Perseverance: 6 Holtoff-Wethey, altered to Edwards duplex, roasting 21,000 tons sulphide ore per month.

The Kalgurli G. M.: 9 Edwards simplex, and 6 Edwards duplex, roasting 11,100 tons sulphide ore per month.

The Associated G. M.: 4 Edwards duplex, roasting 10,800 tons sulphide ore per month. ½

South Kalgurli: 10 Mertons, roasting 9700 tons sulphide ore per month.

Associated Northern: 6 Mertons, roasting capacity of 3600 tons sulphide ore per month; but now on customs ore which varies considerably.

Kalgoorlie Gold Recovery Co., Ltd.: 1 Edwards simplex and 1 Edwards duplex, roasting 500 tons sulphide ore per month. Now shut down.

Chaffers G. M.: 4 Edwards duplex, roasting 2000 tons sulphide ore per month. Now shut down.

Golden Horse-Shoe: 3 Edwards duplex, roasting 2000 tons concentrate per month.

The Ivanhoe: 5 Edwards simplex, roasting 1950 tons concentrate per month.

The Oroya-Brownhill: 3 Mertons, roasting 1000 tons concentrate per month.

Lake View Consols: 3 Edwards simplex and 1 duplex, roasting 1800 tons concentrate per month.

Hainault: 2 Edwards simplex, roasting 600 tons concentrate per month. Now shut down, as mine was acquired by South Kalgurli.

ROASTING AT KALGOORLIE

By M. W. von Bernewitz

(May 13, 1911)

At the Associated Northern Blocks, Kalgoorlie, Western Australia, the ore from the Main Lode is of a rather soft schistose character and contains tellurides and pyrite, the sulphur averaging 3%. The ore from the West Lode is much harder, containing practically no telluride, but some graphite and 4% S. All the ore is crushed in Krupp ball-mills through a 27-mesh screen, and fed into six Merton ordinary furnaces. These have three hearths 6.5 by 30 ft. each in the clear, and a finishing floor 6.5 ft. in diameter. The furnaces are run at one revolution in 45 seconds, and absorb 2.5 hp. each. It was found that, with the first mentioned class of

*Figures revised by the Editor.

ore, 120 tons per day could be roasted; but with the latter, the tonnage fell off to 110 and 100. The furnace feeders, simply small screws in a cast iron tube at the top floor of the furnace, and driven by sprocket wheels and chain, are aranged for different quantities. The Merton furnaces here are fairly satisfactory, but at times difficult to regulate. On the top hearth the ore is merely warmed. Roasting commences on the second, while on the third hearth most of the sulphur is eliminated, helped, of course, by the action on the finishing floor. The temperature is not easily regulated, and the third floor is depended upon to do too much work. The flue-gas averages 600° F., a mercury pyrometer being fixed in each goose-neck flue that connects with the main and stack, the latter 100 by 6 ft., bell-bottomed. The draft averages 0.5 inch in the main flue, dropping to 0.02 inch at the fire-box. There being so many turns in the furnace, the draft is checked somewhat. If

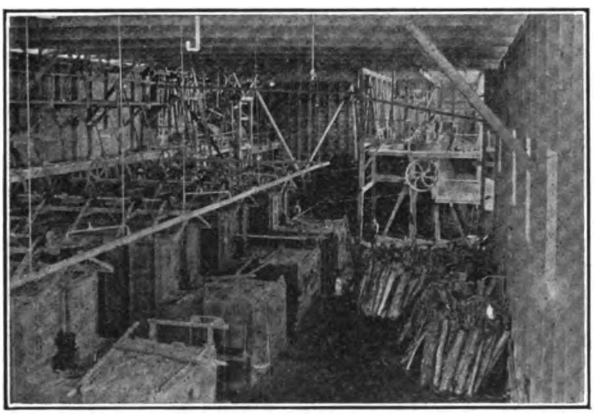


FIG. 1. INTERIOR OF ASSOCIATED NORTHERN MILL, SHOWING MERTON FURNACES.

the furnace was fired too heavily, the ore tended to 'ball,' and the roast would test poor. This 'balling' is a peculiar thing in roasting in Kalgoorlie ores, and was discussed at one of the meetings of the Australian Institute of Mining Engineers. When the heated ore 'balls,' it does not get hard, as the slightest touch with a bar will break it up again. One metallurgist thought that it must be the result of a change in the lime in the ore during roasting to $CaSO_4$. Another said that possibly the schistose nature of the ore would account for it. However, it is rather interesting, and if the lumps are discharged from the furnace, they enclose particles of ore not properly roasted; hence, it is desirable to prevent their formation.

On firing up afresh, it is found advantageous to let in an extra quantity of air for perhaps one-quarter hour, until the firewood is well alight, there being a deficiency of oxygen at this time, due to the new fuel. It is not customary to fire continuously, as tests showed that, if this were done, the soluble sulphides in the roasted

ore were much higher than when firing was intermittent. Roasting the heavy sulphur ore after it had been crushed through a 30-mesh screen was tried, but the result was little better than with the coarser material.

In connection with the subject of furnace design, namely, superimposed hearth furnace and the straight-line type, the remarks of J. E. Edwards in 1909 before the Western Association of Technical Chemists and Metallurgists should be read. At the Associated Northern, the formation of sulphuric acid has been noticed for years, on the arches under the furnaces where the footstep bearings of the rabble spindles are. This acid is highly concentrated, and is less noticeable toward the finishing hearth. Its formation was puzzling for a long time, but it is probably due to gas drawn down between the spindles and bottom hearth, and condensing on the cool arch outside. Fuel consumption averages 14% of weight of ore fed into the furnaces. The discharged ore is taken by a short push-conveyor to an elevator, and from this into a mixer, and mixed with 0.04% KCN solution. The resulting pulp flowing to the pans is very hot. There does not appear to be any extra consumption of cyanide on this account, and fully one-half of the gold in the ore is in the solution at this point, so much so that, when roasting is good, no more cyanide is added to the agitators, which only work for four hours. For a quick, though rough, test of the roasted ore, the lead acetate method is used; while for accurate work in determining soluble sulphides the iodine test is used. The cost of roasting averages 66c. per ton roasted.

No accurate estimate of the amount of dust saved was ever made, and as for determining stack losses, that is rather difficult. The dust is raked out at regular intervals into a small push-conveyor, and is elevated to the fine-ore bin, mixing with the raw ore from the mills to be roasted. This mixture roasts fairly well, but the half-roasted flue-dust tends to make the ore flow very fast in the furnaces. The dust from the ball-mills was drawn off by a fan, and blown into a canvas-lined house for collection. This stuff was exceedingly fine, less than 200 mesh, and roasted well. In the furnace it puffed up a good deal, and gave the impression of a tremendous feed coming through. The ball-mill dust is now blown into the furnaces on the second hearth. Part of the Northern mill is set apart for custom work now, and some interesting lots of ore come in for roasting. Perhaps the most interesting lot was one of 40 tons of concentrate containing over 20% arsenic. One Merton furnace dealt with nine tons daily of this, giving a splendid roast. As soon as the concentrate was fed into the top hearth it caught fire, and the sulphur and arsenic were well out before the end of the third hearth. Dense white fume of arsenious oxide was emitted from the stack, and quickly settled to the ground, but there were no complaints.

The ore at the Associated mine is hard and high in sulphur, the average during the month these notes were written being 5.5%. The ore that comes from Tetley's section of the mine runs as high as 7% S.

The ore is crushed in Krupp mills through an average screen of 28 mesh, and the fine ore is fed into 17 ordinary Merton, one Associated, and two Edwards duplex furnaces. The Merton furnaces were built badly at the start, and reconstruction has not helped them much. Being generally in poor order, their capacity on this ore is only from 10 to 15 tons each daily. Four of them were experimented with on the following lines. On one the finishing hearth was extended about 7 ft., this making three rabbles working in the hottest part of the furnaces; on another furnace the finishing floor was extended to admit putting in an extra rabble; the third was rebuilt similar to the first, only that it had a stack of its own to test the draft; while on the fourth a fire-box was built at the back of the second hearth. The ore roasted gets too hot near the feed end, while the work of the other three is a slight improvement over that of the ordinary type.

The Associated furnace, devised by Messrs. Daggar and Bull, is rather interesting. It has two hearths, each 46 by 6 ft., and the feed travels along the top floor to the fire, drops through a port on to the second hearth, and is rabbled from the fire to the discharge end. The fire has no connection with the bottom floor at all. This is not simply a cooling hearth, as when the ore drops on it not more than 50% of the sulphur has been eliminated. Here it burns away until the third rabble from the discharge, when the ore cools off quickly. The furnace has 14 rabbles on each hearth, traveling at 3 r.p.m., the rabble in the fire end going at 6 r.p.m. it has a stack of its own, with 0.55-in. draft. The stack for this furnace is erected close to the discharge end, there being only a flue of a few feet. With such a strong draft, a fair amount of dust results. A screen test of the dust left in the flue, quoted below, shows that practically only coarse particles are left behind.

Mesh.	Per cent.
On 30	14.3
" 40	42.6
" 60	16.3
" 80	7.3
" 100	4.0
" 120	1.3
" 150	1.0
Through 150	13.0

The daily capacity is 27 tons and the roast is rarely poor. Fuel consumption is about 11%. I think that, some years ago, J. A. Greenwalt, of Cripple Creek, also advocated this principle in roasting.

The Edwards duplex furnaces do admirable work on the heavy sulphide ore. The two in use are not exactly similar. The hearth area, common to both, is 11.75 by 107 ft., with three fire-boxes, each 6 by 2 ft., No. 1 furnace has the first fire-box 56 ft. and the second 66 ft. from the feed end, the third being at the discharge end. This furnace has a fall of 0.25 in. per foot, and the rabbles travel at 4 r.p.m. In No. 2 furnace the fire-boxes are 42 and 66 ft. from the feed end. It has a fall of 0.31 in. per foot, while the rabbles travel at 2.62 r.p.m. The draft averages 0.55 in. There

are 52 rabbles in each furnace, 32 of these being water rabbles. Hot water circulating in roasting plants is, for some reason not yet understood, very corrosive. It eats away pipes wholesale, especially the water-pipes in the rabbles. A ¾-in. iron pipe may last a year, when it is riddled with holes. Copper pipes have been used on some mines with some success. The rabbles used here are cast at the mine, and instead of having a pipe to the toe of the rabble, a web is cast in the rabble about ½-in. thick, and this gets over the trouble.

In the No. 1 Edwards the ore gets a dull red about the fifth rabble from the feeder, while in No. 2 it starts quite near the end, and this furnace does the better work of the two, with about 10 tons more capacity daily, simply because of the first fire-box being nearer the feeder, the extra fall, and slower rabbling. The furnace feed conveyor is fed by a screw, from the fine-ore bin, driven by cone pulleys giving about 75, 90 and 105 tons of ore to each furnace daily. Recently they have averaged about 95 tons each. They are motor-driven, and use 6-amp. at 550 volts each, the ammeter being a splendid guide as to the amount of ore fed into the furnace. These furnaces are easily regulated, and if results are poor at any time, they are stopped for a few minutes, then run slow for perhaps a quarter of an hour, and fired heavily, letting in plenty of extra air. In about half an hour a bad roast is entirely corrected. The end fire-boxes are not used much, just two or three logs are kept burning to warm the air passing through the fire bars. I might say here, with reference to bad roasting, that often this end fire-door is left open to let in an excess of air. It would seem that certain liberties may be taken with such high sulphur ore, much different to an ore carrying only 3% S. The middle fire-boxes are fired heavily, and the sulphur continues to burn until the fourth rabble from the discharge, namely, 22 from the feed end, and then discharges quite cool. Fuel consumption averages 11% of the ore roasted. The flue temperature is 700° F. One man attends two furnaces, and they are greased only on day shift. On these furnaces roasting costs about 60c. per ton, while the total cost for the whole roasting plant is 80c. The roasted ore is elevated and mixed with 0.03% KCN solution, and unless roasting is poor, the consumption is low, and a great deal of gold is dissolved here, the final pulp being agitated for 1½ hours with 0.065% solution in A. Z. agitators.

Since the above notes were compiled, the Associated mill has been overhauled and remodeled, with a view to cheaper and more efficient work. In the roasting department, eight Merton furnaces have been dismantled, and the nine remaining shut down for good, and the flues torn out. In place of these two, two Edwards duplex have been erected similar to No. 2 described, while the fire-boxes of No. 1 have been altered also. The four furnaces now treat 11,000 tons monthly. Except the brick, every part of the roasters was made in the mine's foundry and fitting shop, so they were erected at a minimum cost.

WEDGE MECHANICAL FURNACE

By L. S. Austin

(December 28, 1912)

The Wedge calcining furnace, of the same general type as the McDougall and Herreshoff, has been before the public for some years, and its general appearance is familiar to metallurgists. It differs from the Herreshoff principally in having a large central shaft. The furnace is built in sizes from 12 to 22½ ft. diameter, and with three or more hearths and a dryer hearth. It may be arranged with a fire-box or not, according to the nature of the ore to be roasted. Fig. 1 is a seven and Fig. 2 a five-hearth furnace, the latter having a fire-box.

The ore is fed at the periphery of the furnace at the top or dryer hearth, and is mechanically rabbled across it, entering the

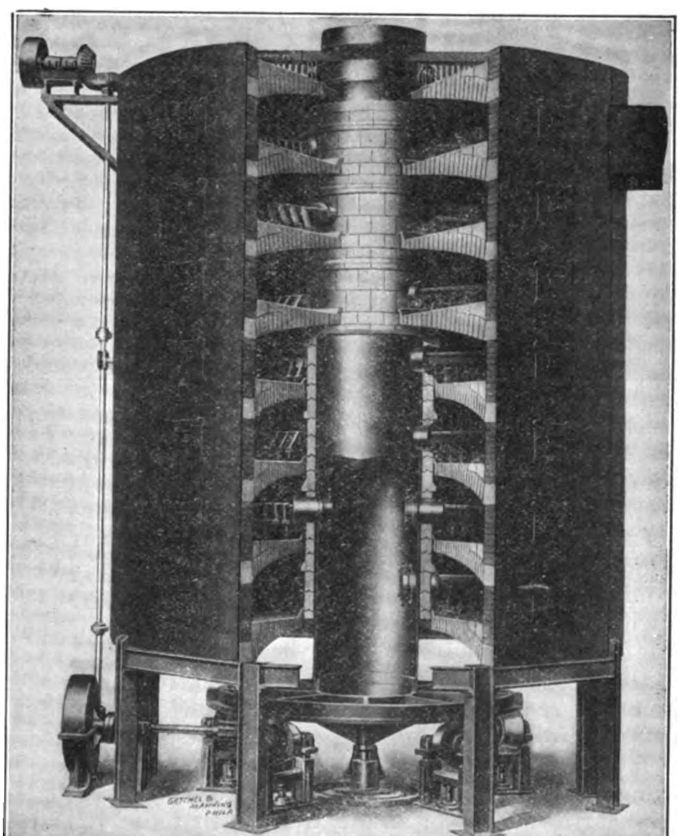


Fig. 1.

upper hearth at the centre of the furnace. By thus utilizing the top of the furnace for drying and pre-heating, the entering ore begins burning on the upper hearth almost at once, and need not wait until dried out. In fact, the heat used in drying out an ore containing 7% moisture may be computed as equivalent to raising the temperature of the dried-out ore to 250° C., at which a self-burning sulphide will begin to ignite.

The central shaft is 4 to 5 ft. diam., and is lined outside with fire-tile, so that it is not so hot inside but that workmen may enter

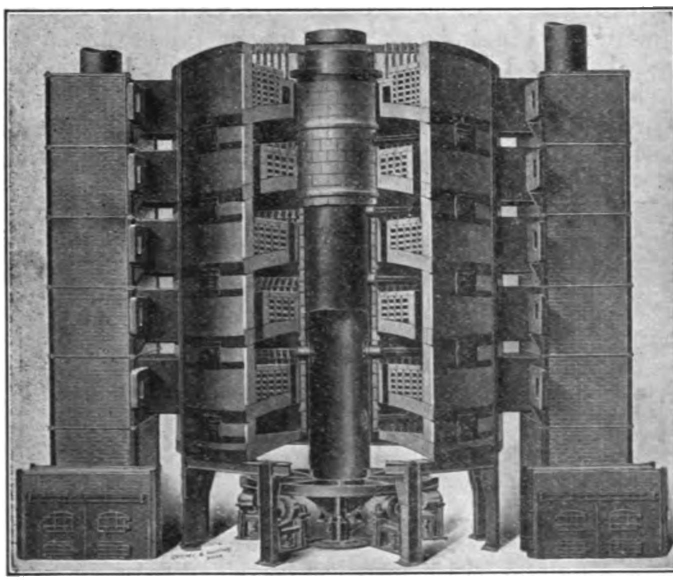


FIG. 2.

at any time and remove the breech-block of any arm that is worn out. Workmen on the outside withdraw the arm through one of the doors and insert a new one, the men inside again replacing the breech-block. Each arm, whether air or water-cooled, has its own supply and discharge pipe, and hence independent regulation and replacement. The weight of the central shaft and attachments is not borne on a step, but on roller-bearings.

The furnace is built with flat firebrick hearths, since accretions are more easily removed from the flat hard surface, which is better adapted for uniform rabbling. Fig. 2, the fire-box furnace, has an external fireplace, so that it can be used for a dead roast or for a chloridizing or sulphatizing roast. The passage, or drop-holes, from hearth to hearth are left of ample area, so that the velocity of the upward passing gases is low, and hence less dust is carried away by the air currents.

AGITATION

MODIFICATION OF PACHUCA-TANK PRACTICE

By Amos J. Yaeger

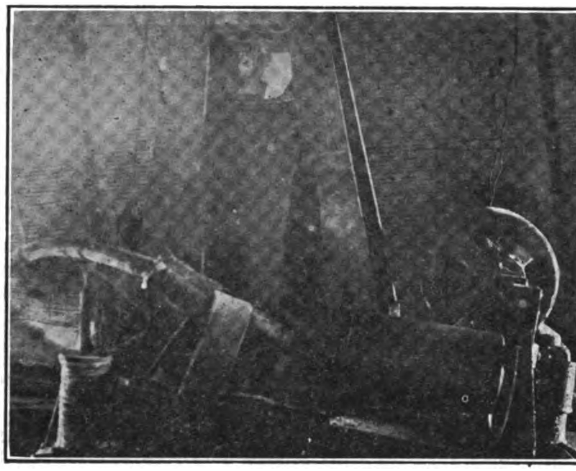
(October 22, 1910)

Having seen various articles in your journal on the subjects of pulp agitation and zinc dust feed in connection with the cyanidation of silver ores, and believing that there are some interesting developments on these lines, I give you the following data: In this plant the usual custom in grinding is followed, namely, preliminary crushing in a rock-breaker and stamps (20), then tube-mills, one 4 ft. 6 in. by 16 ft. Allis-Chalmers mill and one 8-ft. Hardinge conical mill. Classification is accomplished with Dorr classifiers and thickeners, followed by pneumatic agitation in Pachuca tanks, 13 by 55 ft., filtering on two Oliver continuous slime filters, 11 ft. 6 in. by 12 ft., and precipitation with zinc dust in Merrill press. The crushing, classification, and settling of the ores here present no unusual features, but in the matter of agitation and precipitation I think practice here has developed some new points which will be of interest to the profession.

Agitation.—While Pachuca tanks are recognized as a simple, cheap, and to a certain extent, efficient means for the agitation and aeration of pulp, they fall considerably short of what might be desired in the matter of aeration, most of the air escaping in large bubbles at the top of the column pipe with no benefit to the pulp. Observing this, efforts were directed to overcome the defect by cutting off the column pipes as follows: Out of three Pachuca tanks the column pipe in one was cut off at one-quarter distance from the top and raised, all the air thereby being brought into more general contact with the pulp from a point one-quarter distant from the top of the tank, and a reduction in the value of tailing became immediately apparent. The column pipe in No. 2 Pachuca was then cut in two in the middle and the upper half raised, resulting in still better aeration and a still further reduction in value of tailing. Pachuca No. 3 was left with column pipe full length. With the tanks in this condition several charges were run in each, the average tailing in No. 1 being 0.05 oz. per ton lower than from No. 3 and those from No. 2 being 0.16 oz. lower than No. 1, or 0.21 oz. lower than No. 3, the unaltered tank. The theory being thus apparently confirmed, a second series of five charges per tank was run with the column pipes as above described (No. 1 one-quarter off, No. 2 one-half off, No. 3 full length), resulting in average tailing as follows: from No. 1, 1.90 oz. per ton, from No. 2, 1.84 oz. per ton, and from No. 3, 2.09 oz. per ton. In all cases after 18 hours agitation. Subsequently all column pipes were cut in the middle and the upper portions raised, resulting in a corresponding reduction in the value of tailing. The cutting of the pipes has the further advantage that it is possible to commence the agitation of a tank when it is only half charged; also agitation is more easily started when for any reason it has been necessary to temporarily

cut off the air; and agitation from the middle of the tank tends to prevent the settling of sand. The maximum power required for agitating a 100-ton charge of 1½ solution to 1 of pulp is 4 hp. Leaving the Pachucas the pulp goes direct to the Oliver filters, which easily handle 200 tons per day between them in a satisfactory manner, considerably exceeding the rated capacity. The solution is then precipitated by the use of zinc dust in a Merrill press, to which it is fed by gravity flow.

Precipitation.—The zinc was originally fed with the Merrill zinc dust feeder, consisting of a 12-in. belt conveyor, 10 ft. long, operated by floats, and discharging into a cone, where emulsion was supposed to be made by the use of air. Owing to the irregularity of the operation of the belt and the uneven distribution of zinc dust on it, and the fact that the cone failed to make a perfect



DEVICE FOR HANDLING ZINC DUST

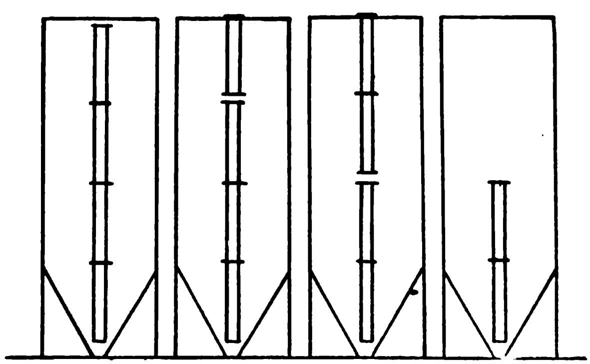
emulsion, a large percentage of the zinc settling and caking on the sides of the cone, the whole apparatus was abandoned, and the following equipment made, which is now in use. This equipment consists of a receiver having a diamond-shaped cross section (as described by J. S. Colbath in *The Engineering and Mining Journal* of February 26, 1910), this taking the solution direct from the press, measuring the flow, as indicated by a register attached to a tripper, and also regulating the quantity of zinc required. The receiver operates the tripper above mentioned, which makes an electric contact each time the receiver dumps. The contact current passing through a coil-magnet operates the zinc-feeder, which has a ratchet attachment for the purpose, and which is situated at the clarifying tanks, 400 ft. distant from the press. The feeder is operated on the Challenge principle, the feed of zinc being further regulated by a

gate on the hopper. The zinc dust is discharged into a launder and flushed direct into a miniature tube-mill, 4 by 14 in., charged with pebbles and run at a speed of 60 r.p.m. This makes a perfect emulsion, which we find indispensable; has materially reduced our consumption of zinc, and resulted in producing a high-grade precipitate, running 80% silver, whereas formerly our precipitate ran only from 35 to 50%. This scheme has been in operation several months with excellent results. It is automatic, and the cost of operation is practically nil, the tube-mill being operated with a ¼-hp. motor.

(December 24, 1910)

The Editor:

Sir—In view of inquiries received for further details of the modified Pachuca-tank practice as carried out at the plant of the Zambona Development Co., which I described in your issue of October 22, I send the following: In all cases of cutting of the column pipes, the upper portion was first raised about eighteen inches, thus allowing most of the pulp to escape at the point cut, some of the pulp continuing to flow upward to the top and over



1. Normal position of column pipe.
2. First alteration made.
3. Second alteration.
4. Final form adopted and used.

the raised portion of the pipe. Noting the beneficial effects of the change, the raised portions of the pipes was removed entirely from all the tanks. The accompanying sketch, with explanations at foot, will, I think, make the whole operation clear.

AMOS YAEGER.

Minas Nuevas, Sonora, Mexico, November 15.

AIR-LIFT PUMPING

By Edward A. Rix

(October 15, 1910)

*It is still a popular fancy that air in lifting fluids from depths acts in a great measure on an ejector principle and all sorts of nozzles and cones are designed to take advantage of this supposed action of compressed air, but it is all much simpler than that, and the basis of the lift action of air lies in the fact that the discharge pipe contains a mixture of air and water which weighs less than the solid water and which surrounds the discharge pipe; consequently the heavier surrounding water pushes the enclosed lighter mixture upward causing the phenomena known as 'air-lifting pumping.'

Most pumping experiments by this method lie within the limits of 125 ft., consequently the commercial tables, curves and data in general have been calculated for such conditions, and these will be helpful in consideration of deep well pumping.

In discussing air lifts certain general terms are used and must be understood. By lift is meant distance from the surface of the liquid being pumped to the point of discharge. By submergence is meant the depth of the discharge pipe under the surface of the liquid being pumped. By percentage of submergence is meant the ratio of the length of the submerged portion of the pipe to the total length of the discharge pipe. The total length of the discharge pipe will of course be the lift plus the submergence. For example, if the surface of the water be 100 ft. below the point of discharge this would be called a lift of 100 ft.; if the discharge pipe extends below the surface of the water 150 ft. it would be called submergence of 150 ft., the total length of the discharge pipe would then be 250 ft. and the submergence would be called 60%. If there be a given ascertained percentage of submergence the actual submergence may be ascertained by multiplying the lift by the percentage of submergence and dividing this product by one hundred minus the percentage of submergence, expressed as follows:

$$\text{Submergence} = \frac{\text{100 per cent} - \text{per cent of submergence}}{\text{Lift} \times \text{per cent of submergence}}$$

Thus in the above example,

$$\frac{100 \times 60}{100 - 60} \text{ or } \frac{6000}{40} = 150 \text{ submergence}$$

From the beginning of air-lift experience it was assumed that the most economical condition for air-lift pumping was when the submergence was 60%, but recent developments have somewhat shaken this idea and it is doubtful whether there has been enough

*Abstract from *The Oil Industry*.

of accumulated data collected on this subject to make any definite statements.

The first formula used for determining the amount of free air required to do pumping, assuming 60% submergence, was as follows:

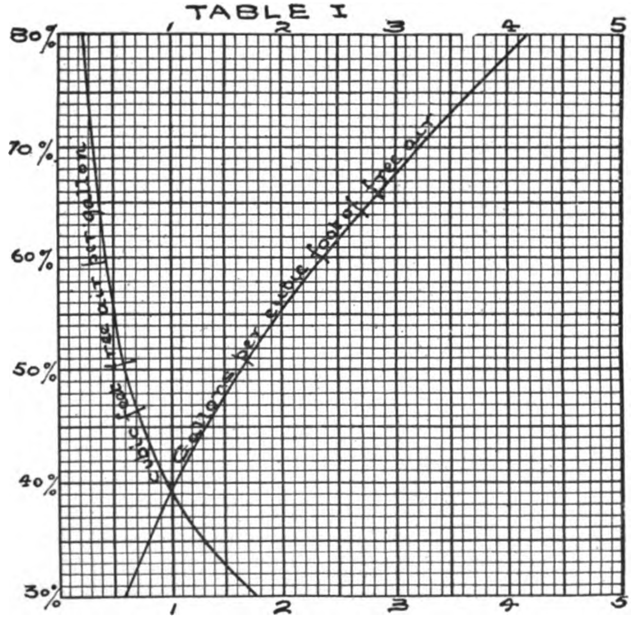
$$\text{Quantity of free air required} = \frac{\text{Gallons and lift}}{125}.$$

This is a rough rule which still holds good for small lifts, up to 100 ft., but it is too generous for deeper pumping. For example, if it be desired to know the amount of free air required to pump 100 gal. per minute 125 ft. high the result would be:

$$\text{Quantity} = \frac{125}{100 \times 125} = 100 \text{ cubic feet}$$

of free air, and the pressure required would always be measured by the submergence; thus in the above problem the lift being 125 ft. and the submergence 60%, the submergence would be 1½ times the lift or 187 ft., and the working pressure would be that due to 187 ft. Strictly speaking this would be about 80 lb., but inasmuch as there is pipe friction to be considered it is safe to take this pressure in pounds equal to one-half the submergence, thus one-half of 187 is 93½ lb., which would be a safe working pressure for such conditions.

TABLE I



When compressed air is introduced in the well in a finely divided state so that the bubbles are small and evenly distributed throughout most of the water the best results are produced. It is evident that if the air pipe merely discharges the air into the water with the full opening of the pipe the result will be large bubbles instead of the finely divided condition which is desired. This has led to the construction of many different patterns of what are called 'pump heads,' which is another name for the extremity of the compressed-air pipe fashioned in such a manner as to distribute the air to the best advantage to the water being pumped.

The Indiana Air Lift Co. issues an interesting diagram, which I have marked table No. 1, and the table of dimensions which I have marked table No. 2. These may be considered fairly accurate at the lifts from 10 to 125 feet.

With these two tables it is easy to determine all the elements of an ordinary air-lift installation for pumping water. Having determined the percentage of submergence by dividing the submergence by the submergence and lift—let us say it was 50%, look along the left hand edge of table No. 1 and find 50% and follow along to the right until it intersects the second curve and you will have the value 1.65 which means that at that submergence one cubic foot of free air will lift 1.65 gal. of water.

TABLE II.
Capacities of Air Lift Pumps

Air Pipe	Size Pump	70%	65%	60%	50%	40%	33%	Size of Well
⅜	1	15	10	10	9	8	6	2½
½	1¼	30	15	15	13	10	8	3
¾	1½	40	25	25	22	15	10	3¼
¾	2	75	55	45	40	30	20	4
1	2½	100	80	75	65	50	30	4⅝
1	3	200	120	110	95	70	50	5⅝
1	3½	225	150	150	125	90	70	6
1	4	350	200	200	170	125	90	6⅝
1¼	4½	450	275	250	210	170	120	7⅝
1¼	5	550	350	320	260	200	150	8
1½	6	800	500	450	380	300	200	9⅝
1½	7	1200	750	650	550	425	300	10⅝
2	8	1700	1000	800	675	525	400	11⅝
2	9	2000	1250	1050	900	700	500	13
2	10	2500	1600	1300	1100	900	600	14

Table No. 2 gives the proper sizes of the Indiana pump heads and the proper sizes of water and air pipes for any given condition. Following up the previous problem of 50% submergence, it it be wished to deliver 125 gal. of water per minute the 50% column should be followed down to 125 gallons, then on the second column will be found the size of the Indiana pump head, namely 3½ inches, which is also the size of the discharge pipe. On the first column will be found the size of the air pipe, 1 in., and on the extreme right hand column will be found the smallest size well that will contain the outfit; namely 6 inches.

It will be noted that no mention is made of the lift, because the table being intended for ordinary conditions of 125 ft. or less, it has been assumed that it takes the same number of cubic feet of free air to lift 10 gal. with 60 ft. submergence as 10 gal. lifted 80

TABLE No. III.

APPROXIMATE CUBIC FEET OF FREE AIR AND WORKING PRESSURE REQUIRED TO RAISE ONE GALLON OF WATER BY AIR LIFT.

$$\text{FORMULA}=\text{Log.}\frac{H+34}{34}\times 234$$

H=Submergence in Feet

L=Lift in Feet

RATIO OF SUBMERGENCE TO LIFT

Lift in Feet	25% 1/3 to 1		33% 1/2 to 1		43% 3/4 to 1		50% 1 to 1		55% 1¼ to 1		60% 1½ to 1		66% 2 to 1		70% 2½ to 1		75% 3 to 1	
	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press	Free air Cubic Feet	Working Press
20							.428	9	.365	11	310	13½	.252	18	.217	22½	.195	27
30							.470	13½	.400	17	.350	20	.290	27	.255	34	.230	40½
40							.508	18	.435	22½	.387	27	.325	36	.287	45	.263	54
50							.546	22½	.470	28	.422	34	.360	45	.320	56	.285	67½
60							.582	27	.510	34	.457	40½	.392	54	.350	67½	.323	81
80							.653	36	.575	45	.522	54	455	72	.410	90	.380	108
100					.850	34	.720	45	.640	56	.585	67½	.512	90	.465	112½	.433	135
120					.915	40½	.785	54	.703	67½	.642	81	.567	108	.520	135	.483	162
140					.982	47	.847	63	.763	79	.697	94½	.622	126	.572	157½	.533	189
160					1.047	54	907	72	.820	90	.755	108	.675	144	.624	180	.583	216
180					1.107	61	.965	81	.875	101	.810	121½	725	162	.672	202	.630	243
200	1.8	30	1.436	45	1.167	67½	1.022	90	.930	112½	862	135	775	180	.720	225	.675	270
250	1.96	38	1.592	56	1.312	84	1.156	112½	1.069	141	.988	168	.895	225	.830	282		
300	2.12	45	1.750	67½	1.507	100	1.292	135	1.206	169	1.110	200	1.010	270	.942	338		
350	2.28	53	1.897	79	1.635	118	1.449	157½	1.317	197	1.232	236	1.110	315	1.050	394		
400	2.45	60	2.045	90	1.725	135	1.542	180	1.429	225	1.345	270	1.232	360	1.155	450		
450	2.60	68	2.182	100	1.850	152	1.669	205	1.542	253	1.457	304	1.337	410	1.280	506		
500	2.74	75	2.328	112½	1.952	169	1.790	225	1.657	281	1.570	338	1.442	450	1.356	562		
550	2.88	83	2.455	128	2.105	185½	1.907	247½	1.772	309	1.678	271	1.564	495				
600	3.02	90	2.564	135	2.225	205	2.018	270	1.884	337½	1.784	410	1.642	540				
650	3.16	98	2.730	146	2.345	219	2.175	292½	2.012	365½	1.908	438	1.742	585				
700	3.31	105	2.845	157½	2.460	236	2.258	315	2.100	394	2.000	472	1.842	630				
750	3.45	113	2.970	169	2.576	256	2.353	337½	2.225	424	2.080	512	1.935	675				
800	3.60	120	3.095	180	2.690	270	2.465	360	2.320	450	2.190	540	2.063	720				
850	3.72	128	3.215	191	2.800	287	2.570	382½	2.410	478	2.29	574	2.125	765				
900	3.85	135	3.338	202½	2.915	304	2.675	405	2.50	506	2.325	608	2.220	810				
950	3.98	143	3.455	214	3.0	322	2.780	427½	2.610	535	2.485	644	2.310	854				
1000	4.11	150	3.575	225	3.465	337¼	2.885	450	2.710	562¼	2.580	675	2.407	900				

ft. with 120 ft. submergence, the working pressure only changing, in the former case being 20 lb. and the latter 40. Now this assumption is not exactly true, but within the practical limits of these lifts it is near enough to be a good convenient rule. When more accuracy is required for greater depths table No. 3, calculated by George H. Reichard, is valuable, as it takes into consideration the expansion of the air bubbles on their way from the lower depths to the surface.

The reason these expansions must be taken into consideration is evident from the very nature of the action of the air lift. Inasmuch as the action of the air lift depends upon an emulsion of air and water, which mixture is lighter than water, it is evident that a perfect condition would be where the bubbles, when introduced at the bottom of the well would maintain the same size in their passage to the discharge. It will readily be seen, however, that inasmuch as the pressure is relieved from the air bubbles as they rise toward the surface, the bubbles get larger and larger, the proportion of air to water increases exactly in proportion to the expansion and this decreases the efficiency of the lift.

The quantity of air given in this table, No. 3, is 2½ times the theoretical quantity required to do the work. Two and one-half has been selected as a co-efficient in this matter as a result of experience. Some engineers have advocated the use of 3 and even 3½ as a co-efficient, but I believe the table as given to be approximately correct. In the first line of the table the percentage of submergence is given; also the ratio of the submergence to the lift. After having determined the amount of free air necessary to do the pumping and the pressure required, then by reference to table No. 4 the brake horse-power to compress the air may be

TABLE NO IV.
Brake Horse Power to Compress 10 Cubic Feet Free Air Per Minute.

Gauge Pressure	Brake H. P.	Gauge Pressure	Brake H. P.	
5	.235	130	2.14	
10	.435	140	2.23	
15	.606	150	2.31	
20	.756	200	2.60	
25	.9	250	2.85	
30	1.02	300	3.07	TWO
40	1.25	350	3.26	STAGE
50	1.45	400	3.40	
60	1.60	450	3.54	
70	1.77	500	3.68	
80	1.92	600	4.00	
90	2.05	700	3.85	
100	2.18	800	4.00	THREE
110	1.98	900	4.16	STAGE
120	2.07	1000	4.32	

determined. This table shows the actual horse-power necessary to compress 10 cu. ft. of free air per minute to the pressure mentioned. An allowance is made in this table for friction and other losses of power, and is generous enough to allow an ample amount of power to do the work.

HISTORICAL NOTES ON THE AIR-LIFT AGITATOR

By J. W. SWAREN

(September 30, 1911)

Early in the year 1880, the Tonite Powder Co., a British corporation, built a powder factory in Contra Costa county, California, to supply the Pacific Coast States with 'tonite,' a safety powder made from gun-cotton and barium nitrate. To wash the gun-cotton and remove the last traces of acid a pneumatic agitating device, identical in theory and similar in design to the agitator now known as the Pachuca or Brown tank, was built. The original idea came from England and how long it had previously been used in the English factory of this company I have been unable to ascertain.

Referring to Fig. 1, accompanying, the general dimensions and appearance of the tank will be seen. It was built of redwood, like a truncated pyramid, with the smaller base down. It was 16 ft. high, the upper base being 8 ft. square and the lower base 3 ft. As first constructed no central tube was used. Air under a pressure of 12 to 15 lb. per square inch was admitted into the bottom through a nozzle, and agitated the mixture of gun-cotton and water with which the tank was filled to within about four feet of the top. This tank was operated with fair results until about 1882, when William Letts Oliver, the manager for the Tonite Powder Co., conceived the idea of using a central tube, as shown in the illustration. A tapering tube 18 in. at the top and 12 in. at the bottom, 7 ft. long, was constructed, with its lower end 2 ft. above the bottom of the tank. The nozzle of the air-pipe was extended to reach just inside this central tube. The agitation was very much improved by this tube, and was found to be best when the larger diameter of the tube was at the top. After this tube was introduced the charge would settle in the bottom when agitation was stopped to permit decanting, and the air did not have sufficient force to set the charge in motion again. To overcome this a jet of air was admitted through a pipe entering at one corner, as shown. This effectively broke up the cake and started the charge. This tank is still in use for agitating gun-cotton at the works of the California Cap Co., which purchased the plant of the Tonite Powder Company.

In the late eighties, Mr. Sommers, of the University of California, discovered that sulphur bichloride (SCl_2) had the extraordinary property of changing the nature of animal oils, and the discovery was believed to be of great commercial value. However, the reactions set up were of such violence that difficulty was experienced in getting a proper mixture. He consulted Mr. Oliver, who suggested using an agitator similar to that described above. In 1890, the American Lucol Co., of which Mr. Oliver was manager, erected a plant at Stege, to manufacture lucol, a composition of fish oil, kerosene, and sulphur bichloride, an excellent substitute for

linseed oil. A redwood tank, 16 ft. high and 8 ft. diam. was erected. The central tube was 18 in. diam., 8 ft. long, and 1 ft. above the bottom of the tank. This tank was filled with fish oil above the central tube, and the air admitted. Sulphur bichloride and kerosene

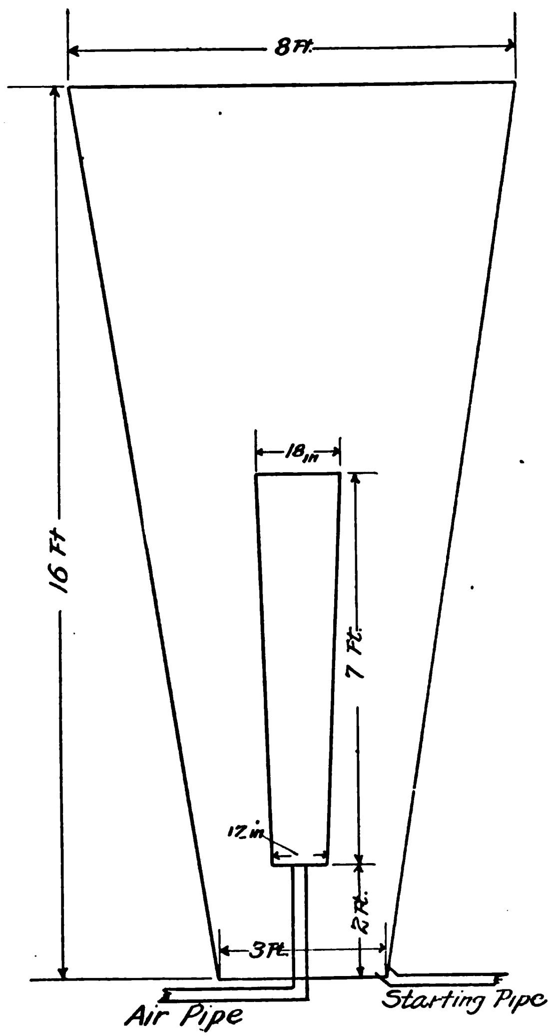


FIG. 1.

were then run in simultaneously from tanks arranged as shown in the sketch. Considerable care had to be exercised in mixing these ingredients, as the sulphur bichloride and fish oil mixed alone made a hard rubbery mass, while the kerosene and fish oil emulsified. To

prevent this, the mixing of the ingredients had to be very rapid and thorough. An air pressure of 12 to 20 lb. was used in the central tube.

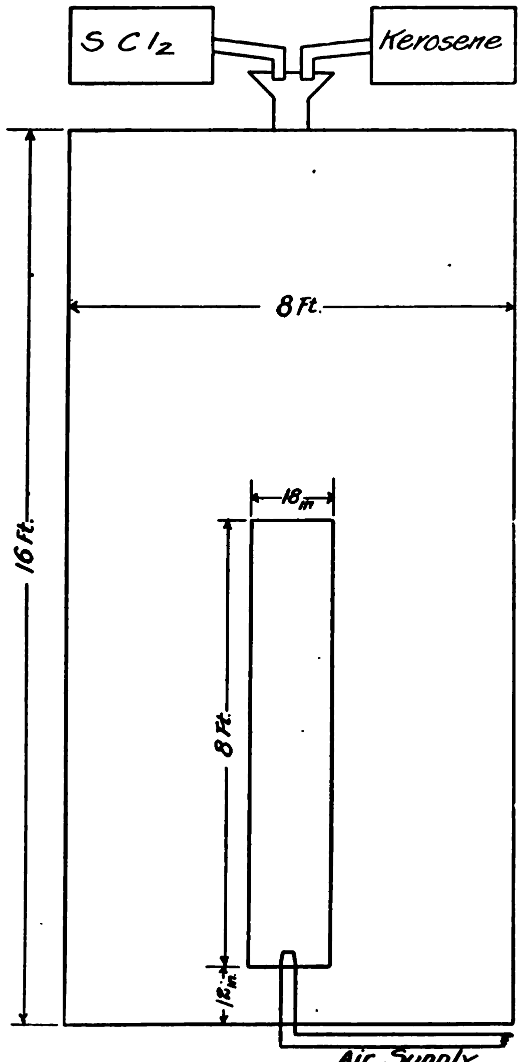


FIG. 2.

The operation of this agitator was so satisfactory that in 1894 four similar tanks were constructed in a new plant for the American Lucol Co., at Carteret, New Jersey. These tanks were steel, with 18-ft. sides and 10 ft. diam., with 16-in. central tubes 12 ft. long,

as shown in Fig. 3. These tanks were used continuously until the American Lucol Co. went out of existence in 1896. A novel means for determining the rate of agitation was employed in this process. Small bits of colored paper were introduced in the tank one color

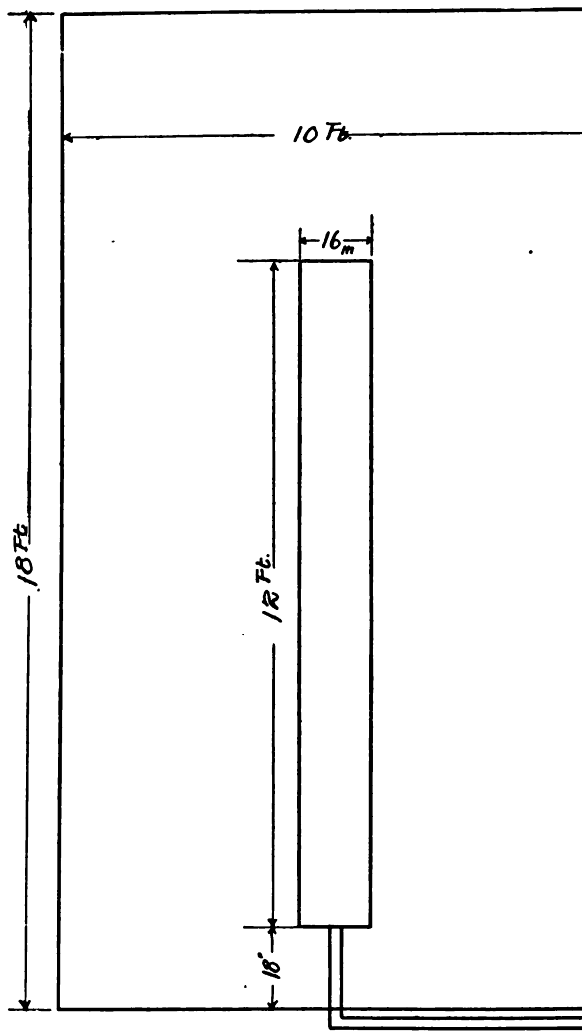


Fig. 3.

at a time, and a sample taken from the different parts of the tank gave a visible indication of the remarkable efficiency of the agitation. These pneumatic agitators antedate the experiments of F. C. Brown at Komata Reefs by some years. According to his contributions to the *Mining and Scientific Press* September 26,

1908, the first pneumatic tanks were installed early in 1902, and the central tube was not introduced until 1904. Pneumatic agitators with central tubes had been employed at the North Star Mines, Grass Valley, California, previous to 1904.

During the summer of 1903, Edwin Letts Oliver, metallurgist for the North Star Mines Co., Grass Valley, California, and son of W. L. Oliver, began experimenting with electrolytic processes for

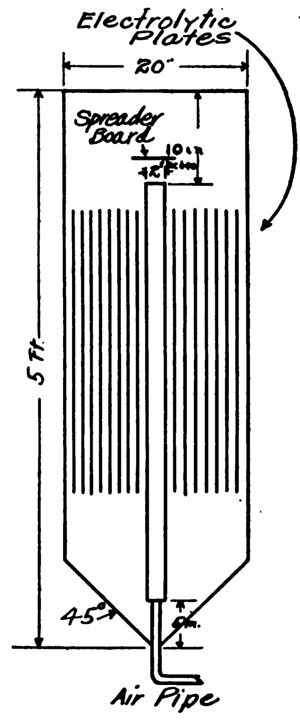


FIG. 4.

recovering the metal content direct from cyanide treated pulp without separating the solutions or using zinc. In this work thorough agitation for the pulp was necessary, as well as some means of bringing the pulp into the electrolytic field of action. A steel tank with 5-ft. sides and 20 in. diam., with a 45° conical bottom was constructed. A 2-in. central tube was used with the lower end 5 in. above the apex of the cone, and its upper end 10 in. below the top of the tank. The nozzle of the air pipe extended into the central tube. A spreaded board was suspended over the top of this tube. The amalgamated copper and iron electrolytic plates were suspended in the annular space, surrounding the upper end of the central tube. Fig. 4 shows the general arrangement of this agitator. This tank was constructed in August, 1903, and proved sufficiently efficient to warrant the construction of additional tanks.

In December, 1903, two additional tanks were built of wood, with 12-ft. sides and 60° cone, and 6 ft. diam. The central tube extended from 1 ft. above the apex of the bottom to about 18 in. from the top of the tank. A spreader board 10 in. diam. was suspended 6 in. above this tube. The first tube used was 3 in. diam., but this did not give sufficient agitation and was replaced by an 8-in. tube. The nozzle was placed inside the tube at first, but was gradually shortened until it was inside the 4-in. discharge pipe,

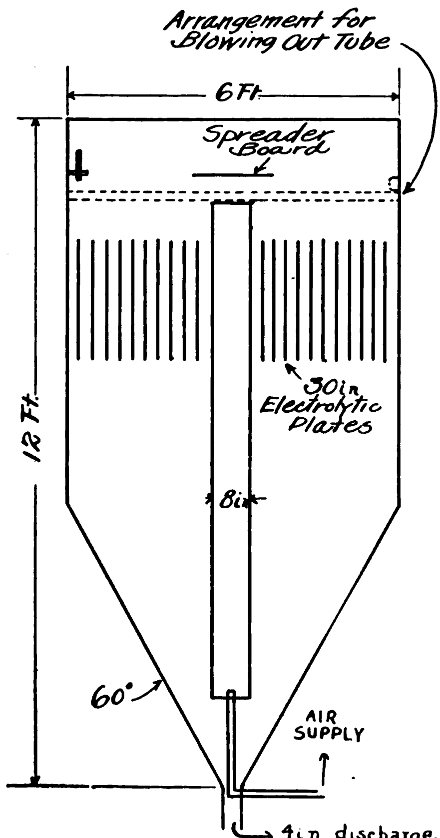


Fig. 5.

and just below the apex of the cone. When the central tube choked, as it did occasionally before the nozzle was shortened, the spreader board was removed and a cover board attached. A lever was placed on top of the tube. This lever was held under a cleat at one side of the tank and forced up tight by a screw device on the opposite side. Air under 80 lb. pressure was then turned into the air-supply pipe, quickly and effectively preaking up the compacted pulp. Fig. 5 shows the general arrangement of these tanks.

A few weeks after these two were completed a third tank was built. In the spring of 1904 twelve larger tanks, six for the North Star and six for the Central cyanide plant were built, 8 ft. diam., 14-ft. sides, with 60° cones and 8-in. central tubes with spreader board and cover board for breaking the pulp when the tube became choked, as shown in Fig. 6. The pulp originally sent to these

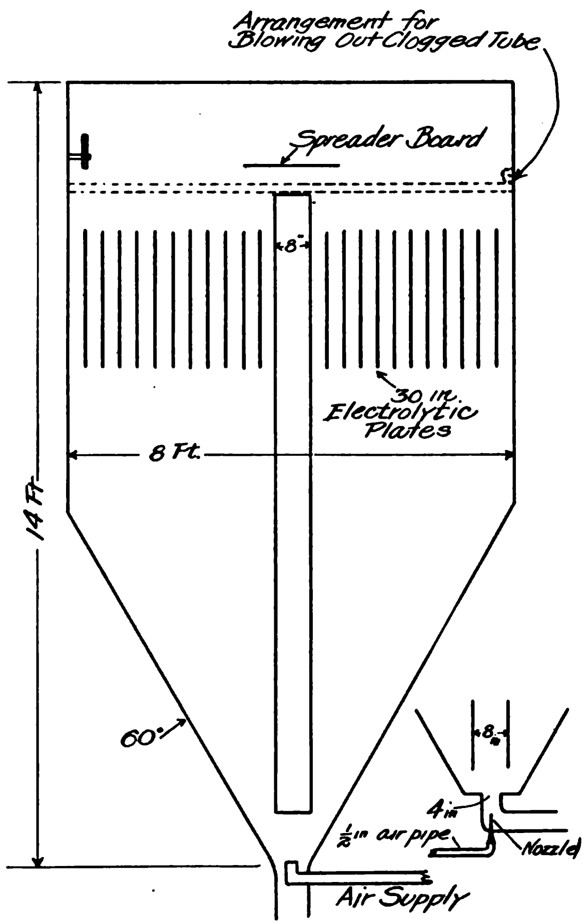


FIG. 6.

tanks was concentrator tailing crushed through a 30-mesh screen at the battery. Later, to increase the capacity of the cyanide plant, the sand was classified out and leached. The electrolytic plant worked as described until January, 1905, giving approximately 75% recovery. This process, which is described fully in U. S. Letters Patent No. 784,120, worked beautifully in theory.

Copper plates were used as cathodes and ⅛-in. iron plates for anodes, 1000 sq. ft. of cathode per agitator being used. The pulp was agitated in a solution containing 0.05% KCN and 0.05% CaO. No salt was needed to increase the electric conductivity of the solution. Contact was made between the cathode and current supply by specially shaped mercury cups, which permitted the mercury to spread over the plate, thus constantly refreshing the mercury surface. The layer of amalgam was formed on the copper plates as in ordinary amalgamating practice. The most economical density was found to be about 50c. gold per square foot. Agitation was intermittent, 6 hours being the usual length of time given to each charge. For cleaning up a smaller tank was used. Once a month the copper plates were transferred to the clean-up tank. The direction of the current flow was reversed, the amalgamated copper plates becoming the anodes and planished iron plates were used as cathodes. Clear cyanide solution was used for electrolyte. In precipitating a current density of 0.05 amperes per square foot of cathode was found to give best results. In the clean-up tank, however, this was raised to 20 amperes per square foot, the object being to strip the copper plates as rapidly as possible. Usually three or four minutes sufficed to strip the copper plate of amalgam, which was then removed and re-amalgamated with fresh mercury.

The amalgam formed on the planished iron cathodes during the clean-up as a thick grayish sponge, ¼ in. or more thick. With such a thick layer of amalgam, more or less would shake off in handling the plates. The clean-up of the sludge in the bottom of the tank usually yielded two or three hundred dollars. When a sufficiently thick layer of amalgam was formed, the plates were removed, the amalgam wiped off, and excess mercury removed by squeezing through canvas in the usual way. On retorting, the bullion ran approximately 400 gold, 350 silver, and 250 copper, when all parts of the equipment were operating to the best advantage. The clean-up could be controlled by the current density, weak currents scarcely affecting the mercury, and depositing copper only on the iron cathode. A stronger current dissolved and deposited some silver and mercury, while a very strong current density formed a deposit high in gold, silver and mercury, but low in copper. Complete separation of gold and silver was never attempted. The most satisfactory working was obtained when depositing an amalgam retorting a bullion of the composition given above. The electrical equipment used for this work consisted of a generator rated at 1200 amp., 5 v., driven by a 10-hp. motor.

When leaching the sand was begun, new conditions were met, as the solutions from the sand vats were precipitated by zinc. Barren solution from the zinc-boxes was pumped into the electrolytic tanks. The zinc in solution was precipitated with the gold and silver. When the plates were cleaned up, and the amalgam retorted, the gold-silver-copper-zinc bullion resulting was a brass, having the color of good gold bars. The weight, however, was light. In one instance the assay value of a beautiful bar was

guessed from $5 to $12 by different employees. The assay showed $3. This bar would have turned any gold brick artist green with envy. The iron oxide, ferrocyanide, and other ferric and ferrous compounds, of which considerable quantities were formed, were at first discharged in the creek. After the plant had been in operation some time, an assay was made of these compounds. This showed a value in excess of $200 per ton. After this all scrapings and rust were saved, even broken and discarded plates yielding $75 per ton. In January, 1905, Acheson graphite was substituted for the iron anodes in the treatment tanks. The same current density was employed. Following the introduction of the graphite, an attempt was made to cyanide concentrate, but the sulphides attacked the graphite, disintegrating it very quickly. The experimental work here described was carried on at the North Star mill. During the summer of 1905 the three original agitators were transferred to the Central mill of the North Star Mines Co., for treating concentrates. In October of that year the Central plant was enlarged by the addition of sand leaching.

Early in 1906 electrolytic precipitation was abandoned. Pneumatic agitation was retained, however, using the same agitators, but removing the electrolytic plates. Decantation in settling vats was employed for separating solution and slime, a number of additional vats being erected for this purpose. During the time of these experiments, about thirty months, over $200,000 was recovered at a good profit. The cyanide consumption was high, running about 7½ lb. per ton of ore and about the same consumption of lime. The 'holdover' was high; more than twice that of zinc-boxes. The anodes, whether iron or graphite, were always troublesome, recovery never checked within 10% of assays (these showing good extraction); while the whole process required extreme care in manipulation.

With a view to finding an economical method for handling the slime, Mr. Oliver, early in 1907, began a series of experiments with filtration devices. The first drum filter was built April, 1907, having a diameter of 20 in., and 12 in. long. This worked splendidly as it was turned by hand, and never operated longer than ten or fifteen minutes without stopping. Assays showed excellent extraction, and the quantity of material handled showed the possibilities, if a machine of large size were constructed.

In June, 1907, the first full working size filters were put in operation. Two filters with drums 6 ft. long and 6 ft. diam. were built at this time. The filters as originally constructed did not work satisfactorily. The cake would not discharge properly and a thin impervious coating would form, preventing a new cake from building. When an attempt was made to remove the discharged or the barren cake by scraping, much difficulty was experienced with the tearing of the canvas. Different schemes were tried with varying success, until the plan of holding the filter-cloth in place by a wire winding was adopted. This method of construction solved the last difficult mechanical problem encountered. This

experimental work was also done at the North Star mill. Early in 1908 four filters with drums 10 ft. diam. and 7 ft. long were built for the North Star cyanide plants. To make room for these machines, three of the agitators described above were removed, and the capacity of the remaining three increased by raising the sides 6 ft.; making the total height 20 ft. The central tube was not lengthened, and the spreader board was retained, as it gives a better distribution of the slime, and throws the heavy sand to the outer edge of the tank.

The accompanying illustrations show the six original agitators, three of which are still operating at the Central mill of the North Star Mines Co. The filters first installed are also working, not a single change in construction having been made. Further descriptions would lead to the discussion of present-day devices, so that these notes are submitted as an indication of a few of the difficulties encountered in the metallurgical advances here recorded. Through the courtesy of those responsible for the work described in this article, I have been permitted to examine correspondence and drawings covering the erection of the apparatus here illustrated and described.

(October 14, 1911)

The Editor:

Sir—In my article entitled, 'Historical Notes on the Air-Lift Agitator,' through an error in typewriting, the cyanide consumption for electrolytic precipitation is given as 7½ lb. This should have been 1½ lb. The consumption now on the same slime is between ½ and ¾ lb., the greater portion of which is mechanical loss due to crushing in water. Were the crushing done in solution, the loss would probably be between ¼ and ½ lb., so that a consumption of 1½ lb. without mechanical loss is obviously too high.

J. W. Swaren.

San Francisco, October 3.

AIR-LIFT AGITATION OF SLIME PULP

(May 6, 1911)

It may not be generally known, but the air-lift for agitation of slime has been in use in the Transvaal for some time, and perhaps it may be of interest to give a brief account of the progress made on the Rand with this process, where, however, the decantation system continues largely in vogue. At a recent meeting of the Chemical and Metallurgical Society of South Africa, Robert Allan dealt with the subject and related the experience and results of trials in several countries and also the progress made during the last thirteen years on the Rand.

It was W. A. Caldecott, writing upon the chemical condition existing in accumulated slime, who showed how the neutralization of cyanicides would be affected by their gradual oxidation through

exposure to the air, and in March 1898, in conjunction with John Kelly, he took out a Transvaal patent, No. 1559, for an air-lift apparatus for the agitation and aeration of slime pulp or other gold-bearing material. This method of sweetening accumulated slime by means of atmospheric oxygen is utilized on the Geldenhuis Deep G. M. Co.'s plant and also on that of the Luipaard's Vlei Estate & G. M. Company.

The first company in the Transvaal on record as using the air-lift in the treatment of accumulated slime was the Nourse Mines, Ltd.; a flat-bottomed tank 30 ft. diam. and 12 ft. deep being used. The Geldenhuis Deep G. M. Co. uses two conical-bottomed vats, each 20 ft. diam. and 20 ft. deep, to oxidize accumulated slime, the vertical depth of the cone being 5 ft. These vats have air-lift pipes 16 in. diam. in the lower portions of the vat, of one-half the depth only. Pipes were formerly used in length equal to the full depth of the vat, and afterward shorter lengths were tried with much more satisfactory results. The slime was pulped with two parts of water in a knife-box mixer and then pumped into the air-lift vats, a charge for each tank being 50 tons of dry slime.

Each vat requires 50 ft. of free air, compressed to 10-lb. pressure, per minute. At the plant of the Luipaard's Vlei Estate Co. the accumulated slime after being suitably pulped (lime and lead acetate being then added) is pumped into a conical-bottom vat of 30 ft. diam., fitted with an air-lift pipe 18 in. in diam., 26 ft. high, and 22 in. above the bottom of the vat. The air supply to the air-lift is delivered through a ¾-in. air-pipe, centrally placed inside the lift pipe. This supply is utilized only when the vat is full and its contents are in agitation, but when the vat is being filled or discharged the air-lift is not operated, the pulp being agitated by air supplied through four ½-in. air-pipes outside the air-lift tube. The pulp receives a total aeration of 22 hours as follows: Filling 9 hours (incomplete aeration), air-lift agitation 6½ hours, discharging (incomplete aeration) 7 hours. The charge of the vat is 150 tons of slime and 300 tons of water and this charge is probably the largest single charge of slime that is agitated anywhere. The vat requires about 85 cu. ft. of free air, compressed to between 25 and 30 lb. per square inch. The small amount of sand, which is always present in accumulated slime, is drawn off from the bottom of the cone, as in the separating cone of the laboratory, and sent to the tailing wheel of the cyanide plant. One of the most useful parts of the treatment vat is a conical diaphragm surrounding the air-lift pipe, whose horizontal base is 3 ft. 4½ in. above the bottom of the vat. This leaves an annular space about 9 in. wide between the diaphragm and the sloping side of the vat, through which the pulp passes to the air-lift. This arrangement entirely prevents the banking up of material around the mouth of the air-lift, and reduces the chance of fine sand collecting on the sloping side of the vat above the diaphragm. In general it enables a vat of larger diameter, and consequently of less capital cost per ton treated, to be used than is otherwise possible.

The accumulated slime, after aeration treatment, is pumped to current slime plant for cyanide treatment. The following average figures from a three months' run show such highly satisfactory results as to warrant more attention being paid to this method of treating accumulated slime:

Value of charge	2.460 dwt.
Value of residue	0.278 dwt.
Extraction	77.7%
Lime consumed per ton (50% CaO)	18.079 lb.
Potassium cyanide per ton	0.233 lb.
Lead acetate per ton	0.121 lb.

The total working cost in connection with the treatment of this slime is 3s. 8.1d., which includes 1s. 5.3d. for collecting and pulping.

The Brown vat is also successfully used for the treatment of sand. At the plant of the Simmer & Jack Proprietary Mines, Ltd., an air-lift vat, 13 ft. diam. and 18 ft. deep, with a central pipe 16 ft. 6 in. long and 10 in. diam., is used for treating 12-ton charges of black sand. The black sand is previously finely ground in a small tube-mill. After the pulp of sand and water is in circulation, lime is added, and later on lead acetate also. Agitation is continued until all soluble sulphides are decomposed, a period of about 16 hours. Cyanide solution is then introduced under presure into the bottom of the charge, and the solution of the gold commenced, periods of agitation, settlement, and decantation following one another alternately until the value of the decanted solution indicates complete treatment. After each decantation, a centrifugal pump, connected with the bottom of the vat, and discharging externally into the top of the vat, is used to withdraw any sand which may have settled at the bottom of the cone below the air-lift and put into general circulation again. The head samples assay about 10 oz. gold per ton, and over 97% extraction is obtained with a cyanide consumption of only 2 lb. per ton.

The air-lift has been in use for the elevation of various liquid materials since 1892; in Western Australia it was used for pumping battery tailing in 1902; and though F. C. Brown, as mentioned before, introduced the central air-lift tube into his tall agitators in 1904, A. F. Crosse actually used the air-lift on the Rand for the experimental agitation of slime in 1903. Mr. Crosse's method of decanting an overflow from an agitation-vat seems to have been useful to inventors. In the Patterson vat this principle is made use of as well as that of the Brown vat, decanted solution, free from sand, being passed outside the vat and drawn through a centrifugal pump. This delivers the pulp again to the bottom of the lift-pipe, taking the place of the air of the Brown vat. The makers give 4½ hp. as the probable power required to agitate 50 tons of (dry) slime in a 15 by 45-ft. vat. By way of comparison it may be noted that in a Brown vat of the same size at the Hacienda San Francisco, at Pachuca, 112 short tons of (dry) 'slime' are

agitated with about 2 hp. Vertical circulation pipes in a Brown vat, with their upper orifices, which are protected by goosenecks, just below the surface of the pulp and lower orifices near the mouth of the air-lift were suggested by W. M. Brodie. Their use was mainly the prevention of choking by heavy sand around the opening of the air-lift.

NEW CYANIDE DEVICE

By LEE FRASER

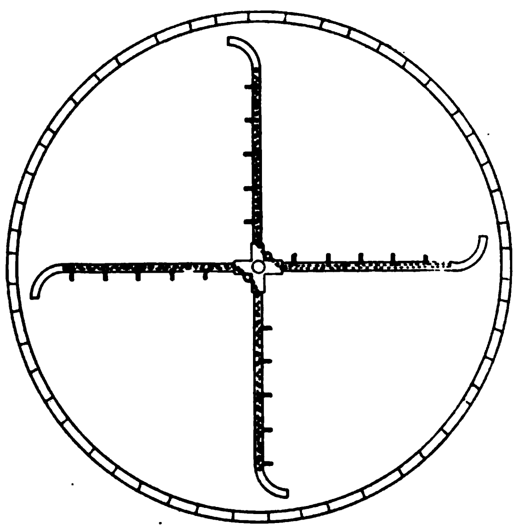
(October 15, 1910)

The rate and extent of the dissolution of gold and silver in cyanide solutions depends almost wholly upon certain variable conditions, the most important of which are: temperature, cyanicides, concentration of solution of cyanide, relative proportion of volume of solution and mass of substance, physical condition of solution and substance while undergoing treatment, oxygen, area of metal exposed to solvent and physical condition of metal to be attacked by solvent. Various machines have from time to time, been contrived, designed to take an especial advantage of one, or a combination of several of the above conditions, to effect a complete dissolution of the precious metals contained in the substance under treatment. To this number, already extensive, I propose to add another. The function of the agitator in slime treatment is to create and to maintain a physical condition of the solution and substance, most advantageous for the progress of the chemical reactions in dissolving the gold and silver. The variable conditions whose regulation, to give maximum rate and extent of dissolution of the gold and silver, lies within the scope of the agitator, will generally comprise: temperature, physical condition of solution and substance undergoing treatment, oxygen, and cyanicides. With but one notable exception, the numerous types of agitators have confined themselves to the regulation of the three latter conditions, neglecting the first.

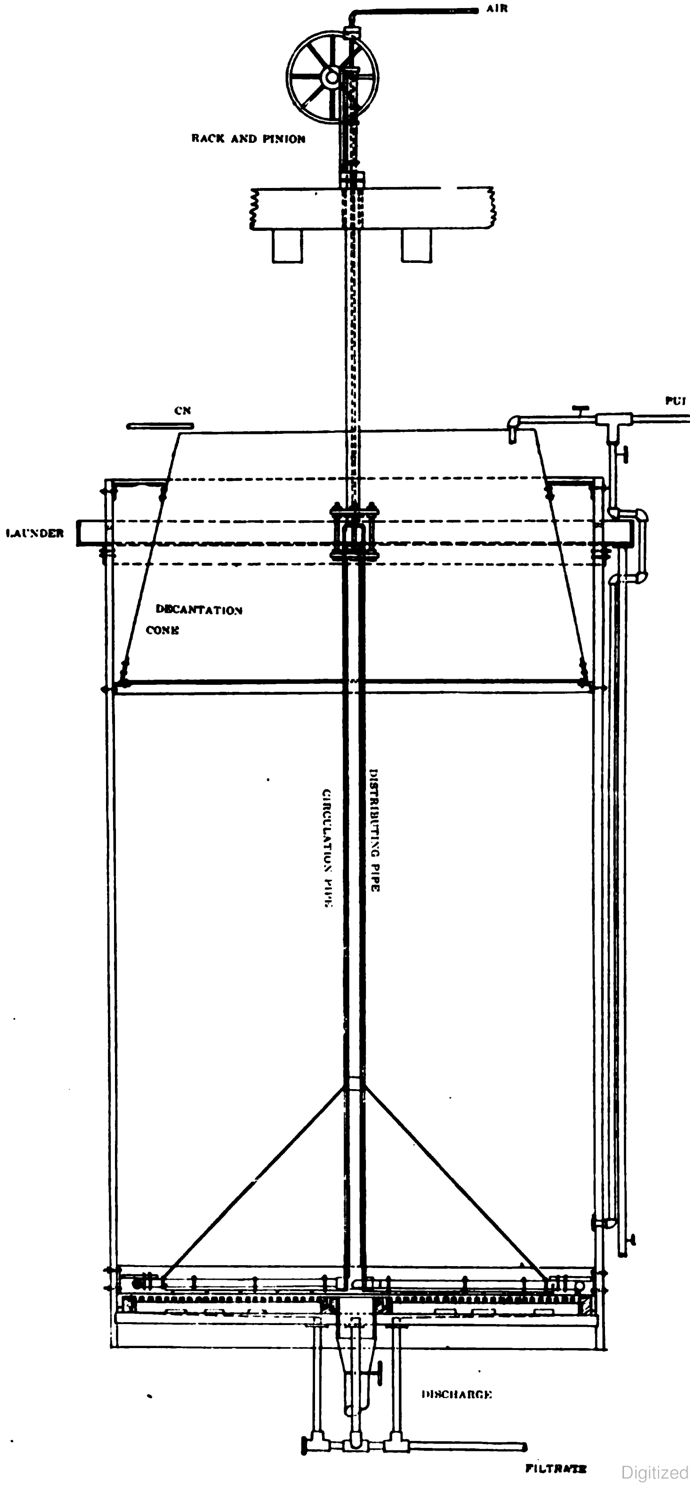
As in any machine, further considerations of efficiency, first cost, and cost of operation, are of vital importance, so that even though a certain machine may make possible a decidedly increased extraction over another, its operation may be prohibited, on account of its failure to compare favorably with other machines, in one or all of these conditions. In the agitator, shown in the accompanying sketches, it has been aimed to regulate only the last three of the first mentioned conditions, and to provide an efficient, economical, and inexpensive machine. It has been designed to also perform the offices of pulp thickener and collecting tank, in which capacities it greatly simplifies any installation of equipment for the treatment of slimes.

It is contemplated that a cycle of operations will consist in the following: The pulp from the classifier is charged into the tank, preferably through the lower outlet, simultaneously with the admission of air into the distributing pipes. The tank is allowed

to fill until the pulp overflows into the vertical circulation pipe, when filtration is commenced, by gravity; or, if necessary, assisted by vacuum. At this same time the circulating pipes begin to revolve, due to the flow of pulp through them. This flow is caused by the air from the distribution pipes, which are led into the circulation pipes near the openings at the ends of the arms, forcing the pulp in the arm out into the tank, thus producing a difference in hydrostatic pressure. Assisting to produce the revolution of the circulation pipes, is the action of the air from the nozzles of the distributing arms upon the surface of the filter bottom. The action of the air from the nozzles, which are flattened to a fish tail, serves to maintain a clear filtering surface, practically free from settling slime, or slime deposited by suction. The nozzles are spaced as points on a spiral curve, and with the positions, every portion of



the filtering surface receives a cleaning at each revolution of the arms. The flow of pulp continues with the filtration until a point is reached where the consistence of the pulp in the tank is that required for cyanide treatment, when cyanide solution is added and the water in the pulp displaced, before submitting the filtrate for precipitation of the gold and silver. As an auxiliary in thickening, decantation is practicable, by placing an inner lining about the upper rim of the treatment tank. Fresh solution is allowed to flow into the tank at the rate the filtrate leaves it, while air is constantly admitted, producing violent agitation and plentiful aeration, as already mentioned. When an extraction has reached its maximum, the pulp is discharged through the valve gate in the bottom of the tank.


AIR
RACK AND PINION
CN
PUI
LAUNDER
DECANTATION
CONE
CIRCULATION PIPE
DISTRIBUTING PIPE
DISCHARGE
FILTRATE

I have attempted here to give a statement of what this agitator is expected to accomplish. Its apparent advantages; low first cost, low cost of operation, high rate of efficiency, adaptability for use as collecting tank, and slime pulp thickener, simplicity of parts, and celerity and completeness of extraction of the gold and silver, have not at this stage been entirely proved by demonstration on an extended or practical working scale.

SLIME AGITATION AT KALGOORLIE

By M. W. von Bernewitz

(June 3, 1911)

Not so long ago mill-men endeavored to make as little slime as possible, and what was made was considered a nuisance. Nowadays, however, the general practice is to make as much as possible, consequently new processes of agitating and filtering the pulp had to be devised, and are being introduced almost continually. We have the choice of the ordinary, A. Z., Kalgurli, inverted cone, Brown and its modifications, silica sponge, air-lift, centrifugal pump, and numerous other styles of agitators, each with its advantages and disadvantages. Agitation as practised at Kalgoorlie, has never been described in a general way, so these notes may be of interest to mill-men.

The following table shows how the ore is crushed and ground in our mills:

Name.	Manner of reduction to slime.
Associated	Ball-mills and pans.
Associated Northern	Ball-mills and pans.
Gt. Boulder Perseverance	Ball-mills, pans, and tubes.
Gt. Boulder Proprietary	Ball-mills, Griffin mills, and pans.
Hainault	Stamps and pans.
Ivanhoe	Stamps and pans.
Kalgurli	Ball-mills and pans.
Lake View & Star	Stamps, pans, and tubes.
Oroya Links	Stamps, pans, and tubes.
South Kalgurli	Ball-mills and pans.

The product in the dry-crushing mills will average about 87% through 150 mesh, while in the wet-crushing plants it will be over 90 through the same screen.

Each system in use calls for some little comment by way of explanation: The A. Z. agitators were introduced at the Associated mill after a long test with the ordinary type. The roasting used to be poor with some of the furnaces, and as the A. Z. agitator showed as much as 4% better results than the latter they were erected. As regards time of agitation, power consumed, and loss of cyanide, it is a case of 1 to 1½ hours, 12 hp., and 1 lb. KCN per ton, against 2 to 12 hours, 2½ hp., and about 1½ lb. KCN respectively, to which must be added the increased extraction. These

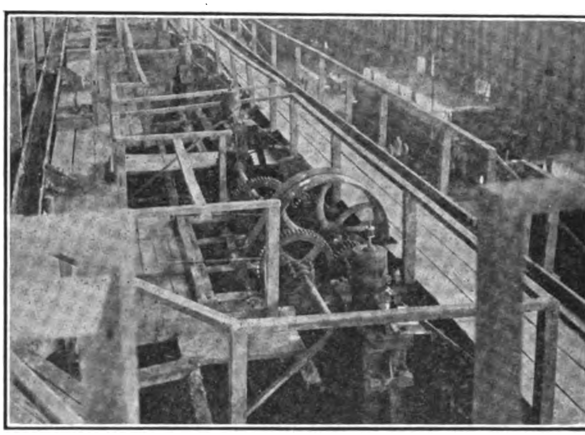
vats are small, but of course the gear may be made for any size tank.

The three propeller blades are fixed on a vertical, gear-driven spindle, revolving at 60 r.p.m. The spindle is arranged for upward and downward thrust by having ball-bearing in the head-frame, from which it is suspended. The blades are set in an angle of about 25°, and work in a 6-ft. ring, made of plate about 12 by ½ in., bolted to the bottom of the vat, and standing about 6 in. above the bottom in order to allow the pulp to circulate to the sides. Directly under the propeller, a false-bottom is bolted to the vat to prevent the sandy particles scouring the bottom. The gear and head-frame stand on 10 by 6-in. H-iron bolted across the top of the vat, and from this are suspended baffle-plates which extend nearly across. The propeller is not started until the vat is half full of slime, and it may be stopped for any time and re-started, the bottom being scoured clean. This is one good point in the A. Z.; a very sandy pulp rarely prevents its starting. When full of slime, the flow from the periphery to the center is at the rate of about 3 ft. per second, the pulp thereby being in violent agitation, and all parts thoroughly exposed to the air. This is where the main advantage of the A. Z. lies, and other ordinary types fail. It may be thought that the consumption of cyanide would be high on account of such violent agitation and exposure to the air; but this is not so, and, providing the solution is in good order, even though the roasting is poor, it will keep up its strength well, as the following tests show:

	Per cent KCN.			
At start of agitation..........	0.072	0.062	0.066	0.064
After ½ to 3 hr. agitation....	0.062	0.056	0.060	0.052

A peculiar point worth notice is that slime which has been agitated for four hours or so, shows at times a higher residue than that which had only one or two hours. Of course, being a gold-bearing ore, solution is affected much quicker than if it contained silver; and it was pointed out to me by an American mill-man that the A. Z. would not pay in agitating silver ore, this requiring up to 90 hours in a Pachuca tank. This is no doubt due to long contact with the air used. Would not a fast-running machine of the A. Z. type expose the pulp to the air sufficiently in a shorter time? In adding cyanide to the A. Z., it is not broken up at all, but lumps of as much as 30 lb. weight are thrown into the slime without damage to blades or gear. Agitation at the Associated Northern calls for little note. If the roasting and subsequent grinding is good, the agitators do good work, and an ordinary circulating solution of 0.04% KCN is sufficient for the solution of the gold, this strength being made up to 0.08% before passing through the extractor boxes. At times extraction falls off, and blowing compressed air in different parts of the pulp has been tried, but somehow it is not of much benefit.

The ordinary agitator acts like a buddle, in that coarse particles settle in the bottom, due to concentration. This concentrate is shoveled out occasionally, and assays as high as $25 per ton, part of the value being in the form of fine amalgam washed over from the pans. This is dried, sent to the mills, and re-treated. The Perseverance has a very large agitation plant doing good work. The ordinary vat calls for no comment; but Mr. Wright, the metallurgist, has devised a new scheme for more rapid circulation of the slime. At an angle of 45°, from the central spindle toward the side of the vat, fixed to the stays of the agitator arms are several pipes up to 10-in. diam., with their tops almost submerged below the surface of the pulp. By the centrifugal force imparted, the pulp is forced from the bottom of the pipes to the top, and flows out in a good stream, thus ensuring that the slime at the bottom of the vat is well mixed with the top section. Mr. Wright believes that



A. Z. AGITATORS AT THE ASSOCIATED

it only takes a few minutes with large pipes to circulate all the pulp in a vat.

The ordinary practice is followed at the Great Boulder. The Hainhault has devised a collecting and thickening vat, the slime from this is circulated by a Frenier pump, and then flows over a sort of cascade for aeration to an agitator of the ordinary style, prior to filter pressing. With the ordinary type, the Ivanhoe agitates its slime with bromo-cyanide. Air agitation is in use at the Kalgurli in tanks somewhat similar to the Brown system, only that they are smaller, and there is no central lift. This system has been in use for nearly ten years, a considerable time before the latter came into use. The tanks are 12½ ft. deep. The first 7 ft. is 6 ft. diam., and from that point the remaining 5 ft. tapers off to 18 in. at the bottom. For the air, there are two 1-in. pipes fitted near the bottom of the tanks, and a little air at 35-lb. pressure is

TABLE GIVING DETAILS OF AGITATORS IN USE

NAME	Type	No.	Size, Ft.	Capacity Tons Dry Slime	Method of Agitation	Speed r.p.m.	Time Hr.	H. P.	Gear	Strength of Cyanide %	Cost Per Ton
(1) Associated	A. Z.	6	17 by 6	22	Propeller	60	1½	12	Bevel	0.065	$0.24
(2) Associated Northern	Ordinary	5	22 by 6	32	Ordinary arms	6	4 to 8	2½	Worm	0.08	0.26
(3) Great Boulder Perseverance	"	24	20 by 4½	15	"	6	2	2½	Bevel	0.07	0.22
(4) Great Boulder Proprietary	"	18	{22 by 5, 18 by 5}	40	"	6	3 to 8	2½	Worm	0.08	
(5) Kalgurli	Air	20	12½ by 6	4½	Air		3½	2	Bevel	0.07	0.21
(6) South Kalgurli	Ordinary	5	21 by 9	40	Ordinary arms	14	4 to 6	2½	"	0.06	
(7) Hainault	"	3	{22 by 7, 22 by 6, 15 by 5}		Frenier pump and cascade	7	4 to 5		"		
(8) Ivanhoe	"	11	20 by 8	53	Ordinary arms		18	3	"	0.08	0.23
(9) Lake View & Star	"	6	20 by 6½	44	"	11	4 to 6	2	"	0.05	
(10) Oroya Links	"	6	17½ by 9	50	"	13	7 to 8	2½	"	0.02	

Note.—Those marked 1 to 6 are in dry-crushing mills; 7 and 8 wet-crushing mills; 9 and 10 are treating old residue. The reason of the much greater capacity in agitators of the same size is due to the extra thick slime run in from the settlers, or from the dumps. The horse-power of an agitator is rather hard to determine from an engine, unless motor driven.

admitted. The pulp first boils quietly. It takes about 20 min. to fill each from the settlers, and about 20 min. to empty by means of a slime-pump which fills the presses, there being a discharge-pipe and valve at the bottom. Each agitator will deal with 24 tons daily.

The details in the table given of the Lake View & Star and Oroya Links refer to the plants re-treating old residue, using the ordinary agitator. Most of the foregoing methods of agitation have been in operation in Western Australia for many years, and apparently give the desired results.

PRINCIPLE OF AGITATION IN PACHUCA TANKS

By A. Grothe

(July 15, 1911)

*In the Pachuca tank, agitation takes place in the body of the tank, during the downward motion of the pulp and does not manifest itself by any vigorous movement. The agitation in the aif-lift is negligible on account of the shortness of the time during which it takes place, and its effect cannot be determined experimentally. The air-lift only serves as a convenient means of aerating the pulp, and lifting the solid particles from the bottom of the tank, after they have fallen through a column of solution equal to the height of the tank. This column itself has a downward motion, caused by the action of the lift. What concerns us is to know the rate of displacement of the solids in the solution.

Calling the velocity of the liquid v, and that of the solids v' we should endeavor to make $v'-v$ as large as possible. As v' is a function of gravity, and the retarding influence by the resistance of the medium, it is a constant in every case, determined by the degree of grinding and the cinsistence of the pulp. The factor v, however, is controllable within certain limits by regulating the quantity and pressure of the air. The greatest displacement of the solids in the pulp is obtained when the current in the lift is just sufficient to keep the cone clear of accumulation of solid particles. Therefore, the less vigorous the circulation the more perfect the agitation.

To determine the value of $v'-v$, a number of experiments were made. An experimental Pachuca tank of 18 in. diam. and 54 in. high, with air-lift, proportioned as in the large tanks, was fitted so that the air-lift discharge could be collected for a short time, instead of being returned to the tank. A charge consisting of a weighed quantity of ground ore and a measured quantity of water was made up, and when normal circulation was established, and sounding showed that no ore had settled near the bottom, samples of about 3 litres each were taken at intervals from the whole of the air-lift discharge, and investigated separately. As the results closely agreed, the average is used in the argument. Charges were made up of three different proportions of solid and liquid as follows:

*From the *Mexican Mining Journal.*

I. 45.5 kg. or with 136.5 litres water, giving 155 litres pulp of 3 to 1.

II. 64.41 kg. ore with 128.82 litres water, giving 155 litres pulp of 2 to 1.

III. 81.30 kg. ore with 121.95 litres water, giving 155 litres pulp of 1.5 to 1, from which the solids per litre of pulp were calculated.

(*a*) I, 294 gm.; II, 415 gm.; III, 524 gm.

The solids per litre of air-lift discharge were: (*b*) I, 426 gm.; II, 501 gm.; III, 580 gm.

The solids have therefore moved at a greater velocity than the water and $v' = \frac{(b)}{(a)} v$. The velocity of the solids was, therefore, I, 1.45 v; II, 1.21 v; III, 1.11 v.

The sizing test of the ore was as follows: All passed 80 mesh; 9.2% was retained on 100 mesh; 27.8 on 200 mesh; and 63% was finer than 200 mesh. A litre of pulp contained, therefore,

	Mesh.	Gm.	Gm.	Gm.
(*c*)	+100	27.05	38.18	48.21
(*d*)	+200	81.73	115.37	145.67
(*e*)	—200	185.22	261.45	330.12

The air-lift discharge consisted of the following sizes:

Mesh.	Per cent.	Per cent.	Per cent.
+100	14.2	12.5	11.3
+200	39.4	32.4	30.3
—200	46.4	55.1	57.4

and the solids contained in one litre of pulp:

	Mesh.	Gm.	Gm.	Gm.
(*f*)	+100	60.49	62.63	65.54
(*g*)	+200	167.86	162.32	175.74
(*h*)	—200	197.66	276.05	338.72

The velocity of the particles of various sizes is,

$$\frac{(f)}{(c)}v, \ \frac{(g)}{(d)}v, \text{ and } \frac{(h)}{(e)}v, \text{ for the}$$

three classes, of pulp, therefore.

Mesh.	I.	II.	III.
+100	2.23 v	1.64 v	1.36 v
+200	2.05 v	1.40 v	1.20 v
—200	1.07 v	1.06 v	1.03 v

The segregation of the particles takes place, not only according to size, but also according to specific gravity. To calculate the velocity of the sulphides, a sulphur determination was made before and during agitation, as follows:

Tank Charge

	I.	II.	III.
Percentage of solids	5.7	5.7	5.7
Grams per litre	16.76	23.65	29.87

Air-Lift Discharge

Percentage of solids	7.7	6.9	6.6
Grams per litre	32.80	34.57	38.28
Velocity of sulphides	1.95 v	1.45 v	1.28 v

The most interesting item of information obtained by these experiments is the displacement of the particles containing silver in the pulp.

Tank Charge

	I.	II.	III.
Grams silver per metric ton....	995	995	995
Milligrams silver per litre pulp.	292	413	521

Air-Lift Discharge

Grams silver per metric ton...	1085	1046	1012
Milligrams silver per litre.....	446	526	587
Velocity of silver.............	1.53 v	1.27 v	1.12 v

So far we have only considered the relative velocity of the solids. To find the absolute displacement, it is necessary to know the velocity of the pulp in the air-lift. Experiments for the construction of empirical formulæ are in progress. It is certain that those for lifts pumping water to a great height are not applicable.

Calling the velocity of the pulp in the lift V, then, the diameter of the lift being one-twelfth of that of the tank,

$$v = \frac{V}{143}$$

Taking V as 100 ft. per minute and the section of the lift as 1 sq. ft., then 100 cu. ft. of pulp is lifted per minute, which contains 88, 83, or 79 cu. ft. of solution according to the degree of density. Taking the last, as the least favorable, v is equal to $\frac{79}{143} = 0.55$ ft. or 6.6 in. per minute. The displacement of the particles in the solution is then:

+ 100 mesh	$0.36\,v = 2.376$ in.
+ 200 mesh	$0.20\,v = 1.320$ in.
- 200 mesh	$0.03\,v = 0.198$ in.
Sulphides in general	$0.28\,v = 1.848$ in.
Particles containing silver	$0.12\,v = 0.792$ in.

Apparently the displacement increases with v and the statement that for efficient agitation, v should be kept as low as possible, seems contradictory. But the experiments were made with the proper amount of circulation. If the latter is increased needlessly, the velocity of the liquid only will be increased and the values found above will become smaller.

The important facts brought out by this investigation, incomplete as it yet is, are: That all the components of the ground ore continually change their position with regard to each other and the solution, thus presenting the whole of their surface to be attacked by ever changing

portions of the latter, leaving behind them a trail of quasi-saturated solution, which is obliterated in the air-lift (if not before) by secondary currents in the tank. The amount of this displacement, even in the most viscous of the pulps experimented with and for the finest and lightest of the particles, is several hundred times their diameter.

AGITATING CONCENTRATE

(July 27, 1912)

The Editor:

Sir—At the Homestake mine at Neal, Idaho, the concentrate is slimed and treated by four to six agitations and decantings in 5 ft. 8 in. by 15-ft. Pachucas. After decanting, a great deal of trouble was met in trying to get the agitation started. The fine concentrate would settle into a cement-like mass in the cone, choke the valves, and resist all efforts to get the air up through it. Incomplete success was obtained by poking the mass with a hand agitator until it was loose enough for the air to break through, but even then the valves and central pipe were usually so clogged that good agitation would not start for hours. Various kinds of check-valves were tried on the air-inlet pipe, but none of them kept the concentrate from backing into the air-pipe when the pressure was turned off.

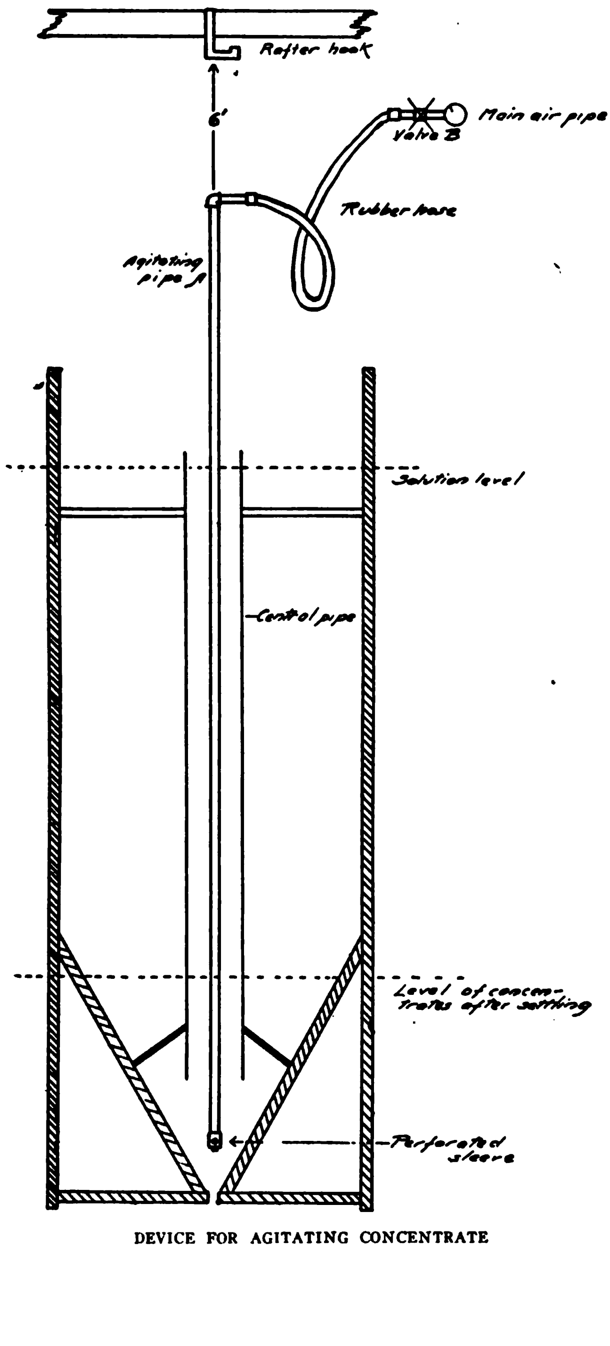
The difficulty was overcome in this manner: a 17-ft. piece of 1-in pipe (*A*) (see illustration p. 350) was fitted with a perforated sleeve and a plug at one end, and lowered, plug end down, into the Pachuca through the central pipe. The upper end was connected by 8 ft. of rubber hose in the main air-pipe. This 17-ft. pipe was raised about 6 ft. and slipped over a hook in the rafters. When it was desired to start the agitation the air was turned on (valve *B*), the pipe was lifted from the hook and dropped. The Pachuca required no further attention until the agitation was complete.

For the first five minutes after the agitating-pipe is dropped, the agitation is confined to within the central pipe, then the weight of the agitator carries it down into the point of the cone and the whole charge is stirred. Finally, in not more than 15 minutes, there is a gush up the centre tube and good circulation is established and maintained. When the agitation is complete the agitating-pipe is lifted, hung on the rafter hook, and the air turned off. When the concentrate settles, the lower end of the agitating-pipe is above the mass, and it is impossible to choke it up.

There is a battery of four small Pachucas equipped with these agitators, and there was not a single case of clogging in six months. This agitator works clear to the bottom of the cone, thus preventing any settling, and after the circulation up the centre is once set up, there is no tendency for the air to escape up the outside of the central pipe. This scheme might be applied to larger Pachucas by operating the agitating-pipe with a chain block.

J. E. ALLEY.

Alta, Utah, June 30.



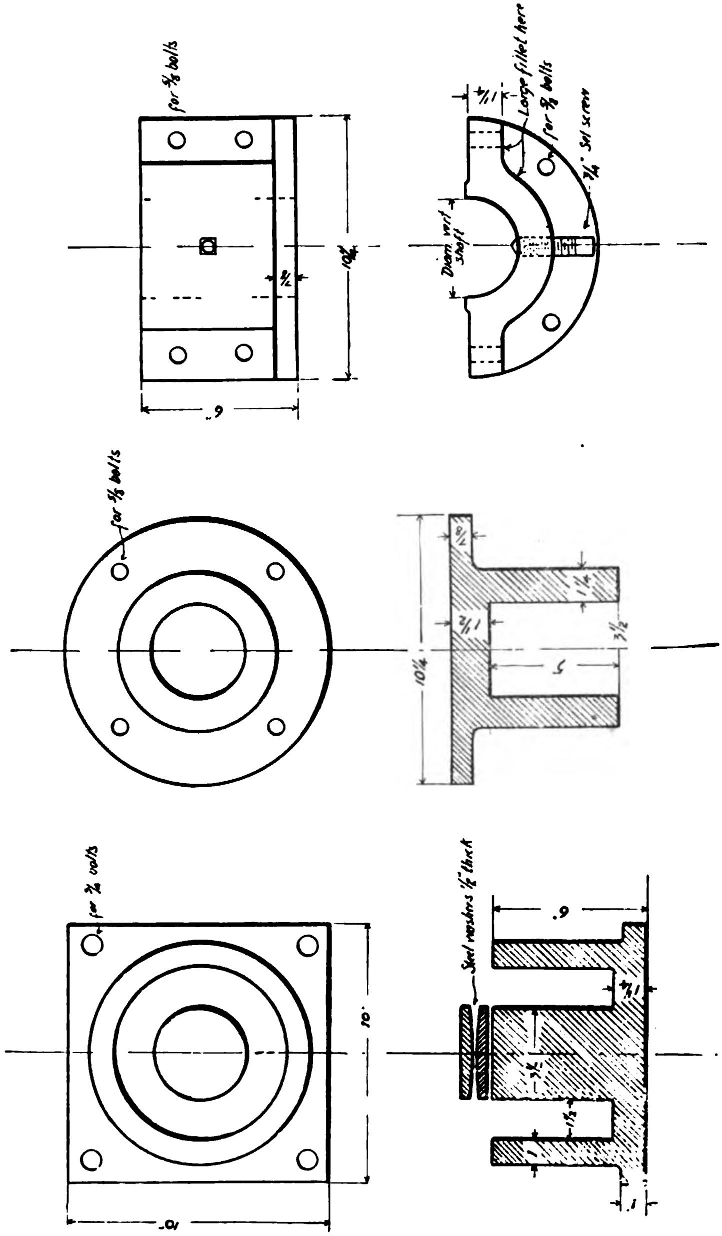
DEVICE FOR AGITATING CONCENTRATE

STEP-BEARING FOR SLIME AGITATOR

By DOUGLAS WATERMAN

(Sept. 21, 1912)

The accompanying drawing shows an excellent form of step-bearing for mechanical slime-agitators in use at the Butters Salvador



AIR-SEALED BEARING FOR MECHANICAL AGITATOR

Mines Ltd. Air-agitation for the treatment of slime is being widely adopted in the more recently constructed cyanide plants, but it would not always be good policy for an old plant equipped with mechanical agitators to make such a radical change in the treatment as the introduction of air-agitation would require. Moreover, the mechanical agitator still has many advocates among cyanide operators, and it will probably never be entirely supplanted by other methods. There is no doubt that it is an effective form of slime agitator. The chief complaint has been the high power-consumption due to the friction in the submerged step-bearing, and the excessive wear in the bearing itself, necessitating constant repairs and renewals.

These objections have been largely overcome in a step-bearing device devised by W. G. Mosher at San Sebastian. I had an opportunity of examining a bearing that had been in constant use for three years without requiring the slightest attention. While the exterior of the revolving parts was worn fully ½ in. smaller than its original diameter by the continued attrition of the pulp, the interior faces were in perfect condition, even the tool-marks being still visible. Only the buttons were worn at the point of contact, but not enough to materially increase the friction. Its form suggests the well known principle of the inverted tumbler in a basin of water, the air entrapped within the cap preventing the rise of the pulp within the annular space around the stem. As an extra precaution against the entrance of the pulp, a sufficient quantity of mercury was introduced to effectually seal the air-space, but it is doubtful whether this is necessary. The two convex faces of the steel buttons form a contact of small area, even after considerable wear, and the power required to overcome the friction is low. It will be noted that the cap is secured to the foot of the shaft by means of the detachable collar. Thus the whole bearing can be removed and replaced without raising the shaft; a distinct advantage. It only remains to add that this device is not patented, and can be turned out by any foundry at small expense.

HENDRYX AGITATOR ON THE RAND

JOHANNESBURG CORRESPONDENCE

(November 9, 1912)

The Mines Trials Committee has been recently testing the Hendryx agitator and an interesting account of the results has been published in the local mining journal, from an authoritative source, but clearly not from the Mines Trials Committee, because, unfortunately, it has decided not to publish the results of the tests.

The following table gives results of actual tests made by the Mines Trials Committee at Wolhuter G. M. Co. plant:

Test No. and date	1—March 11, 1912.	2—March 20, 1912.
Product	Sand and slime.	Slime only.
Quantity and ratio.	Ore, 35 tons; Sol., 70 tons — 1:2	Ore, 35.03 tons; Sol., 70.00 tons — 1:2
Grading:		
+ 60	9.4%	
+ 90	19.3%	0.1%
+ 120	11.3%	0.4%
− 120	60.0%	99.5%
Average assay of charge	2.55 dwt.	1.65 dwt.
Solution strength per ton of ore:		
Start	0.136	0.042
Finish	0.132	0.040
Loss	0.004	0.002
Extraction by solution:		
Hr.	Assay ratio, dwt.	Dwt.
1	1.25	1.0
2	2.00	1.16
3	2.20	1.38
4	2.30	1.52
5	2.30	1.52
6	2.40	
7	2.40	
Extraction by KCN solution	94.1%	92.0%
Washed residue assay	0.281 dwt.	0.05 dwt.
Comparative extraction:		
Residue assay	93.06%	97.0%
Solution assay	94.1%	92.0%

The above figures represent the exact results obtained by the Mines Trials Committee (Johannesburg, Transvaal) in the Hendryx agitator upon residues after amalgamation. The following figures show the total extraction from the original ore, including amalgamation in the mill:

Total extraction by amalgamation and KCN solution:		
Figured from residue	97.9%	99.0%
Figured from solution	97.9%	98.0%

H. Stadler who was in charge of the trials on behalf of the Mines Trials' Committee, states that there are other important facts not published in the results, but which ought to be taken into account. He goes on to explain that in the tests the cyanide had been added in the form of salt, thereby ensuring greater efficiency than by using the ordinary stock solution, while the consumption of cyanide quoted only refers to that used during agitation and excludes losses incurred in the extractor boxes and that going to the dump. The fact that the pulp was heated to a temperature of 90°F. is not made sufficiently

plain in the published results of the tests, while it must not be overlooked that the percentages of extraction were calculated from the values of washed samples. Mr. Stadler further states that he considers the statement published in the official report, that "as much or more of the gold and silver can be dissolved in one hour by the use of the Hendryx agitator as can be dissolved in 24 hours by the usual methods of percolation as somewhat misleading, and explains that in the Hendryx 18-ft. agitator only about 35 tons of mixed sand and slime is agitated during 7 hours, while in every day practice on the Rand, up to 400 tons of sand is treated in 5 to 7 days including the subsequent operations of transfer of sand, settling, washing, re-settling, and emptying of the treatment vat which are not included in the 7 hours mentioned as required by the Hendrix agitator." Mr. Stadler considers that figures relating to the costs should also have been supplied, and goes on to express regret that the results of various tests made by the Mines Trials Committee, particularly those relating to tube milling made during the last three years do not appear. The apparent reason for the non-publication of the results of the tests made by the Mines Trials Committee is the desire that they should only be available to those contributing to the cost.

The Editor:

Sir—I note in your issue of November 9, 1912, an article by H. Stadler under the head of 'Johannesburg Correspondence,' entitled 'Hendryx Agitator on the Rand.' I beg to state that from March 11, 1912, to May 2, 1912, five tests were made, of which Mr. Stadler was only in charge of two. Some of the statements in the article referred to are quite misleading and unfair. When the Mines Trials' Committee was requested by me to test the agitator, their secretary desired for the information of this committee to know what claims were made for the agitator and was informed that it would dissolve as much or more of the gold content of the Rand ore, sand and slime together, in one hour as ordinary leaching methods would dissolve in twenty-four; that the loss of cyanide during the period of agitation would be less than in percolation to dissolve the same amount of gold; that the power would not be excessive, that the agitator would start readily after a long period of settlement; that the slime contained in the ore, owing to the rapid and violent agitation, would be coagulated instead of emulsified, and would more readily settle or filter than when air agitation was used. The official report which the committee has been courteous enough to allow me to publish I hereby enclose.

It is not necessary for Mr. Stadler to give the apparent reason for the non-publication of the tests by the Mines Trials' Committee, as no one is better aware of the fact than he that it is neither the custom nor the policy of the committee to publish results of its tests. It is very necessary that the committee protect itself against misrepresentations and the rule of not publishing such tests, I understand to be a policy of protection to themselves. Knowing that I had come a long distance and had expended a large amount of

RESULTS OF ACTUAL TESTS MADE BY THE MINES TRIALS' COMMITTEE AT THE WOLHUTER G.M. CO.

TEST NO. AND DATE.	1.—MARCH 11, 1912.	2.—MARCH 20, 1912.	3.—MARCH 26, 1912.	4.—APRIL 9, 1912.	5.—MAY 2, 1912.
PRODUCT..	Sands and Slimes.	Slimes only.	Sands and Slimes.	Sands and Slimes.	Sands and Slimes.
QUANTITY AND RATIO ..	Ore: 35 tons Soln.: 70 ,, } 1:2	Ore: 35·03 tons Soln.: 70·00 ,, } 1:2	Ore: 35 tons. Soln.: 71 ,, } 1:2	Ore: 35 tons. Soln.: 72·4 ,, } 1:2·068	Ore: 35·09 tons Soln.: 73·69 ,, } 1:2·1
GRADING: + 60 + 90 + 120 — 120	9·4% 19.3% 11·3% 60·0%	— ·1% ·4% 99·5%	13·75% 17·74% 10·70% 57·81%	5·93% 20.52% 12·0% 61·45%	9·18% 21·10% 11·39% 57·33%
AVERAGE ASSAY OF CHARGE ..	2·55 dwts.	1·65 dwts.	2·2 dwts.	1·95 dwts.	2·3 dwts.
SOLUTION STH PER TON OF ORE— Start Finish Loss	 ·068 × 2 = ·136 ·066 × 2 = ·132 ·002 × 2 = ·004	 ·021 × 2 = ·042 020 × 2 = ·040 ·001 × 2 = ·002	 ·063 × 2 = ·126 ·060 × 2 = ·120 ·003 × 2 = ·006	 ·052 × 2 = ·104 ·050 × 2 = ·100 ·002 × 2 = ·004	 ·0525 × 2 = ·105 ·050 × 2 = ·100 ·0025 × 2 = ·005
EXTRACTION:—BY SOLUTION. 1 hr. 2 hrs. 3 hrs. 4 hrs. 5 hrs. 6 hrs. 7 hrs.	Assay × Ratio. dwts. ·76 × 2 = 1·52 1·00 × 2 = 2·00 1·10 × 2 = 2·20 1·15 × 2 = 2·30 1·15 × 2 = 2·30 1·20 × 2 = 2·40 1·20 × 2 = 2·40	dwts. ¼ hr. ·5 × 2 = 1·0 ½ hr. ·58 × 2 = 1·16 1 hr. ·69 × 2 = 1·38 2 hrs. ·76 × 2 = 1·52 3 hrs. ·76 × 2 = 1·52	dwts. 1 hr. ·67 × 2 = 1·34 2 hrs. ·83 × 2 = 1·66 3 hrs. ·84 × 2 = 1·68 4 hrs. ·9 × 2 = 1·80 5 hrs. ·92 × 2 = 1·84 6 hrs. ·94 × 2 = 1·88	dwts. 1 hr. ·6 × 2·068 = 1·24 2 hrs. ·67 × 2·068 = 1·385 3 hrs. ·68 × 2·068 = 1·406 4 hrs. ·79 × 2·068 = 1·633 5 hrs. ·84 × 2·068 = 1·737 6 hrs. ·84 × 2·068 = 1·737	dwts ·57 × 2·1 = 1·218 ·65 × 2·1 = 1·365 ·79 × 2·1 = 1·659 ·85 × 2·1 = 1·785 ·89 × 2·1 = 1·869 ·98 × 2·1 = 2·058
% EXTRACTION BY KCY. SOLUTION..	2·4 dwts. from 2·55 = 94·1%	1·52 dwts. from 1·65 = 92%	1·88 dwts. from 2·2 = 85·4%	1·737 dwts. from 1·95 = 89·3%	2·058 dwts. from 2·3 = 89·47%
WASHED RESIDUE ASSAY ..	·281 dwts.	·05 dwts.	·35 dwts.	25 dwts.	·55 dwts.
COMPARATIVE % EXTRACTION FROM— RESIDUE ASSAY .. .. SOLUTION ASSAY ..	 93·06% 94·1 %	 97% 92%	 84% 85·4%	 87·2% 89·3%	 76·3% 89·47%

The above figures represent the exact results obtained by the Mines Trials' Committee (Johannesburg Transvaal) in the Hendryx Agitator upon residues after Amalgamation.

OFFICIAL REPORT, MINES TRIALS' COMMITTEE

THE FOLLOWING FIGURES SHOW THE TOTAL EXTRACTION FROM THE ORIGINAL ORE, INCLUDING AMALGAMATION IN THE MILL AND WERE FURNISHED BY THE COURTESY OF THE WOLHUTER G.M. CO.

TOTAL EXTRACTION: BY AMALGAMATION AND KCy SOLUTION— Figured from Residues Figured from Solutions	97·6% 97·9%	99·0% 98·0%	94·5% 94·89%	95·5% 96·2%	91·6% 96·31%

time and money, and that no misrepresentations would or could be made, they had the courtesy to permit me to print the tabulated report above referred to.

The comparison Mr. Stadler attempts to make of my 18-ft. agitator with the 400-ton leaching tank is not only unfair but absurd. Had the committee desired to ascertain how many tons could actually be put through the 18-ft. agitator in twenty-four hours, otherwise than by comparison after the assaying of the solutions, and washing residues at different periods, it undoubtedly would have installed de-watering or thickening tanks as storage ahead of the agitator and decanting or filtering apparatus after the agitator, so the time of de-watering, filling and discharging could be reduced to a minimum; an expense entirely unnecessary in order to ascertain what the committe desired to know. Also, it would have installed a large size 24-ft. agitator, holding 100 tons of solids and 200 tons of solution with proper arrangement for filling and discharging, and would have treated four charges per day of the coarse material, which would have been 400 tons in twenty-four hours as against 400 tons of Mr. Stadler's leaching tanks in seven days. On the slime several times that amount could have been treated.

WILBUR A. HENDRYX.

New York, December 4, 1912.

[Mr. Hendryx is in error in supposing that the article referred to is by Mr. Stadler, as it was sent us by our regular Johannesburg correspondent. In our issue of last week Alfred James speaks forcefully concerning the policy pursued by the Mines Trials' Committee.—EDITOR.]

DECANTATION

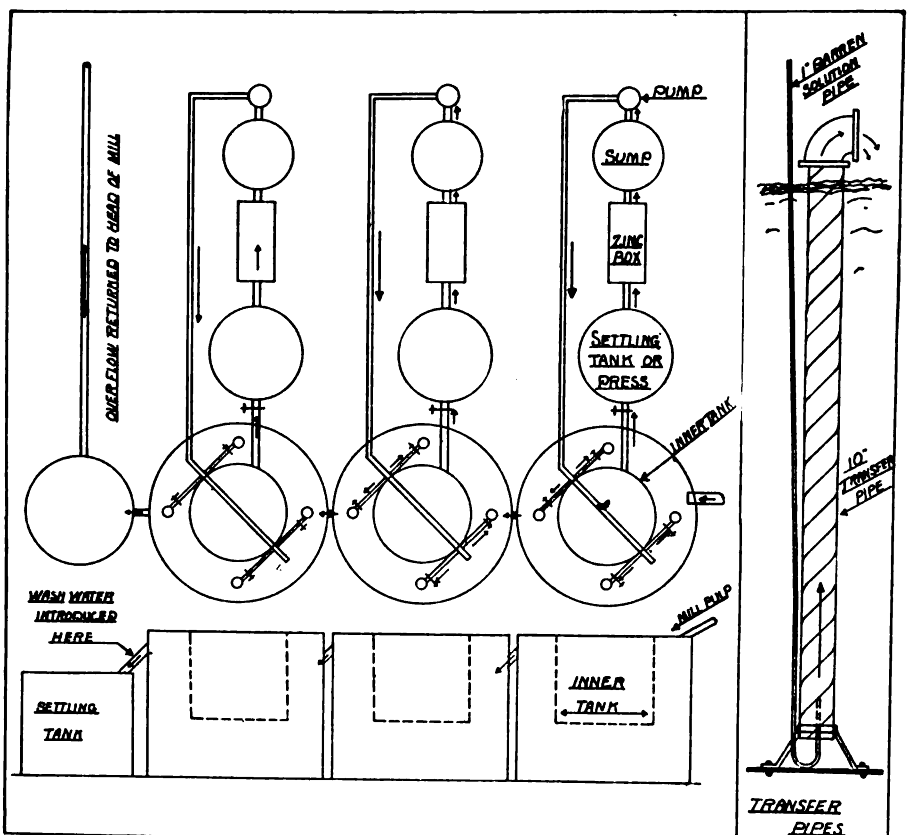
CONTINUOUS AGITATION OF SLIME WITH BARREN CYANIDE SOLUTION

By C. F. Spaulding

(March 1, 1913)

The Ogle Mountain Mining Co., of Oregon City, Oregon, will build a mill this coming summer in which the slime will be agitated by circulating barren cyanide solution instead of air. This mill has been designed from the data worked up at the 350-ton cyanide plant of the Veta Colorado at Parral, Mexico, where the plan worked admirably, reducing the tailing 12 to 15 gm. silver per ton and also cutting the time for agitation from 72 to 48 hr. I believe that the better extraction and shortening of the time was due to the displacement of the pregnant solution surrounding each particle of silver by more active barren solution during each passage through the transfer pipe.

In the figure is shown the flow-sheet of the cyanide plant of the Ogle Mountain Mining Co.'s mill which will be from 100 to 125-



FLOW-SHEET, OGLE MOUNTAIN MILL

ton capacity. The agitating-vats are three in number, 20 ft. in diameter, 20 ft. deep, and have flat bottoms. Each contains an inner suspended bottomless vat, 10 by 10 ft., in which the slime settles. This inner settling-vat is fitted with baffle-plates to stop the swirling motion of the pulp as it rises from the outer vat.

The agitating will be done through four 10-in. transfer pipes made of spiral riveted pipe and fitted with L's on top, these to be four to six inches above the level of the pulp surface and to point normal to radius of vat. The discharge of the pulp through these L's give a circular swirling movement to the pulp, keeping the bottom of the vat fairly well scoured, and preventing rapid settling of the slime. These transfer pipes will be evenly spaced and 2 ft. from the side of the agitating-vat. The solution in the pulp as it settles in the inner vat, is decanted off through a 4-in. horizontal pipe, passing to a clarifying vat or press, thence through zinc-boxes or zinc precipitation-presses and to a sump from which it is pumped back to the agitating-vat in which it is used for agitation. A centrifugal pump is best for this work as the pressures are low, not over 15 to 20 lb. For the Oregon mill, handling 100 to 125 tons per day, tests show that a 2-in. centrifugal pump at each tank, consuming not over 5 hp., will displace the pregnant solution in the vat with barren solution every 6 hr.; equivalent to four complete displacements or washes each 24 hr. Each agitating vat is connected with an independent set of settling or clarifying-vat or press, a zinc-box, sump, and pump. Each unit, being complete in itself, will keep the solution from the agitators from mixing. The mill pulp entering vat No. 1 displaces an equal amount, which goes to vat No. 2, and on through the system. The overflow from the last vat goes to a settling-vat. The settled tailing is discharged and the overflow is pumped back to the head of the mill, where it is brought up to strength and again enters the system. The make-up water for the mill can well be added to the overflow launder from the last vat where it will act as a wash water.

The decanting pipe is proportioned in size to the tonnage in the vat and the time in which it is desired to replace the pregnant solution with barren solution. In the Oregon mill the decanting pipes are to be of 4 in., and the agitating pipes 1 in. diameter.

Agitating with barren cyanide solution has many advantages over air-agitation. For one thing, it is much cheaper. The zinc-box for each tank can be arranged so there is not over 12 to 15 ft. drop between the pulp-level in the tank and the solution in the sump. This will give a small head to pump against and should not take over 2½ to 5 hp. To agitate with air takes 10 to 15 hp. to each vat. The cost of piping and zinc-boxes in each case is about the same. The cost of the settling and clarifying-vats and pumps is small. This is more than off-set by the cost of the filter-press for the tailing which is done away with. In the second place, the system permits easy manipulation. There is only one valve to regulate for each tank. This is placed somewhere in the decanting pipe. The plant will run for days without adjusting this valve. The valves in the four agitating-pipes are not touched after adjust-

ment. The plant is automatic in action and needs little attention. It costs less to operate for labor, power, supplies, and repairs, than a plant agitating with air and including filter-pressing. One man on a shift should be able to attend to a plant up to 300 tons in size. A third important factor is the increased extraction. The recovery will be materially increased and the time of agitation shortened; due, I think, to the operator being able to carry a more dilute solution and to the displacement of the pregnant solution with barren solution. At the Veta Colorado it was necessary to carry a thick pulp, not over 2 to 1 dilution, because a more dilute solution foamed badly. The best extraction was with a pulp 3 to 1 or thinner. With barren cyanide solution this might be as thin as desired without any foaming. The system also saves the cost of an expensive air-compressor and receiver. The agitating pipe is much simpler, being merely turned up at the bottom and using no valves, rubber sleeves, or complicated devices to stop the flow of pulp into the pipe whenever the air goes off. There is no danger of this happening when using cyanide solution for agitation, since the pipe is always full of solution and the pulp cannot back up in it and clog it.

This passage of the pulp through the transfer pipes is due to two things. The barren solution acts as a jet or hydraulic elevator forcing the circulation, diluting the pulp in the transfer column, it makes the latter lighter than that in the vat. This difference in specific gravity also helps the circulation. It results in a particularly satisfactory agitation, with no fuss, foaming, or troubles of any kind. The flow-sheet illustrated includes clarifying-vats or presses. In some cases where the slime contains a large amount of colloidal matter and it is necessary to crowd the passage of pulp, these may be necessary. Usually, however, with sufficient decanting capacity these will not be needed, though it might be well to put them in, to provide extra capacity for decanting. If the clarifying vat or press is used the inner (settling) vat can be made smaller in proportion and the decanting may be crowded, the clarifying device taking care of any colloidal matter which passes over.

Mr. McKinnery, of Ward, Colorado, who has a plant somewhat on these lines, except that he agitates mechanically and keeps the decanted solution passing through the system of tanks against the flow of pulp, finds that the settling is practically perfect. His extraction on heavy sulphide ore is good. By the time the pulp reaches the last vat and is ready to be discharged, the dissolved gold remaining is so small in amount that it does not pay to use a filter-press. Tests show that in the Oregon plant the dissolved gold in the trailing when ready to be discharged will not be over 8½c. per ton. This can be thrown away cheaper than a filter-press can be put in and operated.

An existing plant using Pachucas for agitation can easily and cheaply be modified to use barren cyanide solution for agitation, and it would probably cut out the cost of filter-pressing. In making such a change, build the inner vat 8 to 10 ft. diam., 10 to 15 ft. deep,

and construct it from 1-in. lumber, corrugated iron, or any handy material. It does not have to be especially strong or heavy. Leave the transfer pipe in the centre as usual, cutting into the air-pipe a connection from the barren-solution pump, or, better, putting in a separate pipe for the barren solution. Do the settling outside the inner vat instead of inside it as shown in the flow-sheet. That was the system used at the Veta Colorado mill during the summer of 1911. So far as I know, the process of agitation with barren cyanide solution is not patented and has never been used by anyone except myself. I do not intend to patent the process.

CONTINUOUS DECANTATION

By Donald F. Irvin

(July 22, 1911)

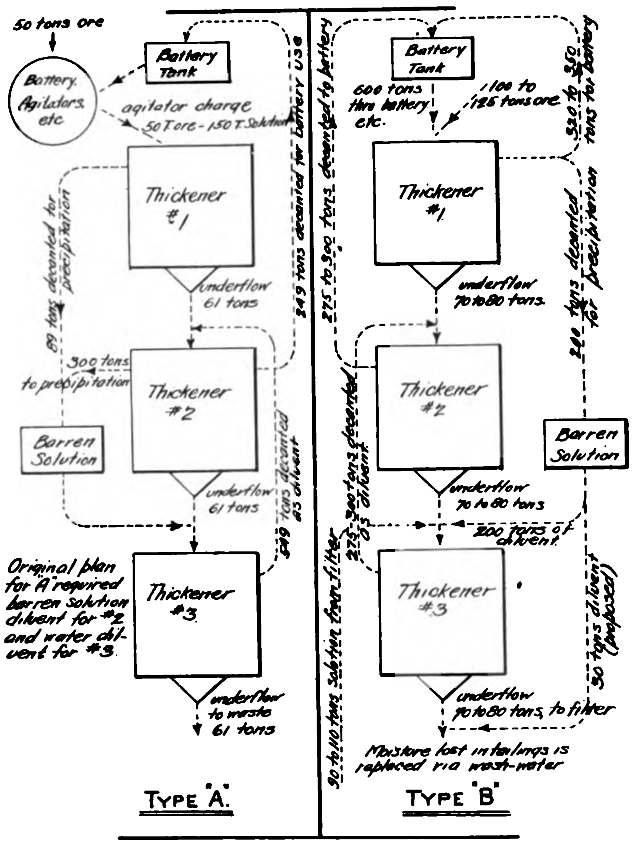
Within recent years there has been a noticeable tendency toward continuous methods in metallurgy. In the cyaniding of slime, which is, in practically every case, accomplished by treatment of successive charges, an excellent field is offered for continuous processes, since the usual three units of the plant, collector, agitator, and filter, receive and discharge pulp intermittently. Furthermore, the widespread use of sliming methods offers a large field for experiment on this point.

I consider a fair standard of continuous operation to be the ability of any unit machine to receive, treat, and discharge its full proportion of mill pulp at all times, without temporarily diverting into another vat any portion thereof at any time. Successful continuous collection and decantation was developed by J. V. N. Dorr in South Dakota, and continuous agitation is well exemplified by the Esperanza mill under M. H. Kuryla. There are other machines than the Dorr thickener and the continuous battery of Pachuca tanks claiming to perform the same work, but it is conceded that these machines were the pioneers.

Using the various types of equipment mentioned, it is now possible, at least theoretically, to feed ore in at the crusher, make a few adjustments, and then recline in the shade, while the plant operates automatically. For two reasons, at least, it is not possible at once to install such equipment in every plant. (1) There is very little information available on this subject, as there are few plants operating which embody the continuous process throughout. (2) Considerations of time and treatment and general suitability of the ore must be observed. If feasible, the continuous process is wonderfully attractive on the grounds of simplicity and cheapness, and with the belief that any information, though incomplete, will be of interest, some data follow, on the plant with which I am familiar, which uses this method.

As a basis of comparison, I have taken a similar scheme of treatment proposed by Ferdinand McCann, which appeared in the *Engineering and Mining Journal*, October 2, 1909. To facilitate

comparison, a parallel diagram of both plants is appended. In both plants the fundamental idea is successive decantation, thickening, and increasing dilution through a tandem set of three Dorr thickeners. I will refer to the plant suggested by Mr. McCann as type A, and the other as type B. Both schemes of treatment require crushing in solution. Type B differs from A, in that the entire mill-flow goes to No. 1 thickener, instead of a thickened charge from an intermediate collector and agitator. This is done because the



FLOW-SHEET OF CONTINUOUS DECANTATION PLANT

ready solubility of the metallic contents obviates necessity for agitation, although for the usual type of ore, an agitating equipment would doubtless be required.

For the foregoing reason, the quantity of solution, taken from No. 1 thickener for precipitation in type A is smaller than type B, but per ton of dry ore treated is nearly the same—1.8 tons in the one case, 2 tons in the other. However, type A requires additional precipitation of low-grade solution from No. 2 thickener—possibly

300 tons. Hence precipitated solution per ton of dry ore treated is 7.8 tons, at least, and McCann advocated a total of 11 tons. In type A, under the original plan, the reduction in value of solutions, between each tank (assuming perfect dilution) is 90%, a figure not attained in B, where figures based on routine operation show a reduction of 40.8% between No. 1 and No. 2, and 43.5% between No. 2 and No. 3.

The average content of the solutions decanted from No. 1 thickener in A and B respectively were 143 gm. of silver and $1.52 gold. The apparently less efficient work of B is explained by the diluent for No. 2 being the decanted solution from No. 3, instead of barren solution, and the diluent for No. 3 being barren solution, of satisfactorily low tenor, but not absolutely barren. Further, a vacuum-filter followed No. 3 in type B, the filtered solution being used as an additional diluent for the pulp flowing into No. 3 thickener.

In an article by L. B. Eames in the *Mexican Mining Journal*, September, 1910, continues decantation is discussed theoretically, the attention is called to possible additional dissolving of metals during the passage of pulp through the tanks. This is shown to be true in B; solution tonnage and assays for the period quoted showing $26 additional dissolved daily in No. 2 and $15.80 in No. 3. A substitution of a fourth thickener for the vacuum-filter in No. 3 would render the plant simpler in operation, but the additional indicated extraction of gold while passing through would be very small. Reckoning from the work of three thickeners it would be negligible; and the grade of solution would be little more than 10c. per ton, and possibly less. A well regulated thickener will discharge quartz slime pulp with 40 to 45% moisture, and a vacuum-filter of the leaf type will filter a cake carrying 30 to 35% moisture, requiring considerable additional water to sluice it away, in most cases probably not less than 50% in all. The value of replacing a fourth thickener by a filter can only be obtained by noting the daily loss of dissolved gold and cyanide in the underflow from the thickener, say 40 tons of solution, carrying 8 to 10c. gold and cyanide, or a total daily loss of about $12, say $4500 yearly, for a 100 to 120-ton plant. Against this can be set down the greater water consumption of a filter-plant (a substantial consideration in many places), the initial cost of the filter, and more rapid depreciation of a filter-plant. Perfect washing of cake is assumed. There is no doubt that a cheap, efficient, and compact filter could be used to advantage, but it must be all that, and with low power and labor requirements besides, for the power, labor, and maintenance cost on a thickener is very small indeed.

McCann makes no reference to the manner of transfer of pulp and solution in type A; this is doubtless the point needing the most care in operation. At plant B decanted solutions were raised by air-lifts and the pulp by diaphragm pumps. The dilution of the thickened pulp can be effectually done by placing baffles in the launder receiving pulp and solution. A small centrifugal pump handles the battery solution, and a small triplex pump raises the

gold solution. In a large plant of this kind, handling several hundred tons of ore daily, units of 100 to 120 tons could be arranged as in type A or B, but with each tank placed successively lower than the preceding one; this would aid in pulp transfer; the limiting case would be when all tanks are discharged by gravity.

Transfer of the thickened pulp at a minimum expense for power and attendance is sought in several ways: one plant used air-lifts, another centrifugal pumps, a third diaphragm pumps, and a fourth discharges into sumps from which the pulp is drawn by bucket-elevators. Regulation of pulp and solution flow is not quickly disturbed when valves are once properly set and pumps are speeded correctly. Barring accidents, type B is a very smooth-working plant. All the work of operation and current repairs is done by one man and a helper—a saving of one or two men over the average slime-plant employing solution-man, filter-man, and pump-man. On the other hand, should an accident occur, it needs immediate attention, and no long delay is admissible. In general, this type of plant is simple in operation, with low labor, power, and maintenance costs, and, when metallurgists generally are familiar with it, should become a popular and standard method of slime treatment.

DECANTING OF SLIME

(Editorial December 14, 1912)

Proposals to revert to decantation for general treatment of slime in cyanidation, are atavistic. The first great step in successful treatment of slime was the application of filtration to the problem. Fine material may be treated by decantation, but filtration introduces notable economies. The reason, as also the limitation to successful application of decantation, is not difficult to see when the behavior of finely ground material in solutions is analyzed. There is more pore space in a fine mud than in coarse sand settled in water, but the spaces between individual solid particles are narrower. It follows that the channels or openings through the mass are smaller and more tortuous. The mass is less permeable, and to obtain the same rate of flow extra force of some sort must be given to the liquid material which it is proposed to pass through the mass of solid particles. In settled sands the open spaces are sufficiently large to permit gravity alone to pull solutions through the mass at a rate that makes commercial results possible in treatment plants. With slime, to obtain the same rate of flow requires a vacuum or pressure.

The proposal to obviate this difficulty by continuous agitation and decantation has been made many times, and plants designed on this plan are in successful operation treating finely ground material that is really a sand. In them the individual particles are kept freely floating, each completely surrounded by solution, and never allowed to completely settle. As a result, there are no pore spaces, and the particle rather than the mass becomes important. The plan

is ingenious and there is a field for such treatment plants, but not, we believe, a universal one. The reason lies in the difference in specific gravity between sulphides and silicious particles. The very ores that most need sliming for successful treatment are those in which the gold and silver is locked in the sulphides. It is to release this gold and silver that the ore is slimed. In practical operation of treatment plants where concentrates are cyanided, it is universally true that the sulphides are found to require more time for treatment than the other material. The relations of the valuable to waste metal are closer and complete solution of the gold and silver is harder to obtain. In continuous agitation and decantation, the first particles to settle are sulphides, and with them goes much of the gold and silver. The result is that the material which should receive the longest treatment really escapes from the system first, and there seems no general way to avoid this without completely destroying the process itself. The old process of decantation is available for coarse material; continuous decantation can be used on coarse and finer material and is useful for taking off the richer solution before vacuum filtration. For slime the best treatment is by filtration, and it is worth noting that the old Australian method, not covered by patent as it happens, of pressure filtration with hand cleaning, is still used. Our New York contemporary sees in decantation the future of slime cyanidation. We hold that for general use it is a backward step, however helpful it may be in special cases or in combination with filtration.

INTERMITTENT SYSTEM OF CYANIDATION

By L. P. Hills

(February 8, 1913)

*The popularity of the continuous system of direct cyanidation is, I believe, an unfair verdict against the intermittent system. The former undoubtedly sprang from the difficulties encountered in the latter in promptly effecting a condition of suspension in the charge to be agitated, clogging of pipes, etc. All of these difficulties can be wholly obviated at present, one means being by the use of the agitator hereinafter described.

Theoretically, the charge system surpasses the continuous. The latter is, by its very nature, inefficient. Assume a series of 100-ton tanks, with a flow of 10 tons per hour. Consider the efflux from the first tank for any given hour. That 10-ton portion is composed of, approximately:

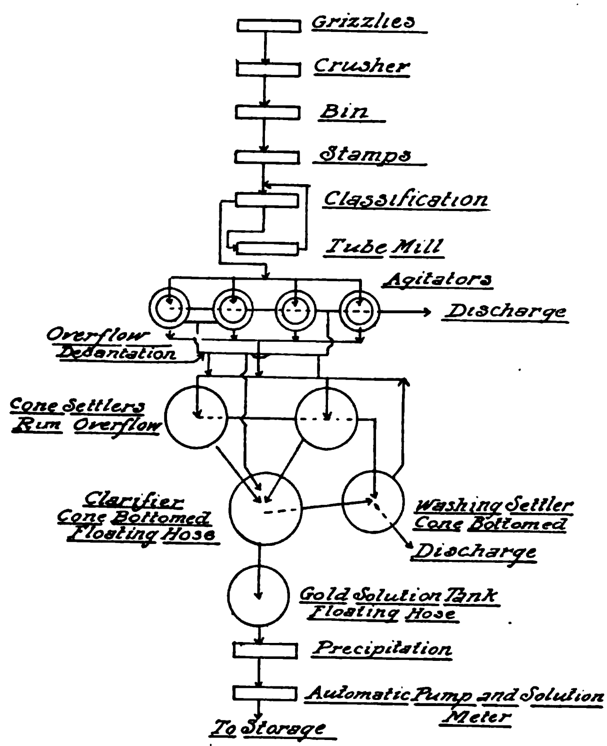
0.909 tons of the influx of the given hour
0.825 tons of the first preceding hour
0.751 tons of the second preceding hour
0.683 tons of the third preceding hour
0.621 tons of the fourth preceding hour
0.565 tons of the fifth preceding hour

*From *Colorado School of Mines Magazine.*

0.513 tons of the sixth preceding hour
0.466 tons of the seventh preceding hour
0.424 tons of the eighth preceding hour
0.386 tons of the ninth preceding hour
0.351 tons of the tenth preceding hour

the remaining 3½ tons in diminishing portions back to the initial inflowing 10 tons. The great bulk of the pulp receives exceedingly long treatment, the time of treatment of different portions varying

Intermittent Decantation System
Flow Sheet
No Filtration



between wide limits. Also the heavier and coarser particles make the most rapid transit through the series; thus that portion requiring the longer treatment gets the shorter treatment. The ideal system, wherein the pulp receives the exact treatment required to perfect extraction, can be closely approximated in the intermittent system.

The accompanying flow-sheet illustrates an intermittent decantation system for which the following merits can be reasonably

claimed; Simplicity, cheapness of installation, economy of power and labor, low maintenance cost, flexibility of operation.

The agitator is the chief feature of this system, and insures reliability of operation. It is a modification of the Pachuca, the central fixed column being surrounded by a vertically adjustable column of a few inches greater diameter. This outer column, by means of a screw at the top of the tank, may be moved from a position where the lower end of the column is in contact with the sides of the cone, to any predetermined point above that, the upper extremity of the column always being below the surface of the solution. During the time agitation is suspended and settling is taking place the outer column is in its lowest position, thus excluding the pulp from settling around the inner column and air-jet. To start agitation, turn the air on, which will institute circulation of solution down between the two columns and up through the central column, whereupon the outer column is raised, allowing the pulp to enter into the circulation. This agitator has been subjected to all kinds of tests and never failed to get the pulp into a condition of suspension and without the aid of additional air.

In the system under consideration, a thickener to precede the agitators is not necessary. The agitator is provided with a baffle which gives a circumferential quiet zone when pulp is being run in and when agitation is taking place, the overflow going to cone settlers. When a charge is sufficiently thickened in the agitator, the pulp is switched to another agitator, and agitation started in the charged tank. During agitation, barren solution may be run in to lessen the gold contents and also to vitalize the solution in the tank.

The extraction being completed, the outer column is screwed down, agitation suspended, pulp settled, solution decanted, and washes repeated as many times as is advisable, depending on the richness of the ore. The settlings from settlers and clarifier are drawn off intermittently to washing settler, or agitator, and washed.

This system permits operating so that the finer pulp, in which the extraction is rapid, receives the short treatment and the heavier and coarser pulp receives longer treatment.

(March 15, 1913)

The Editor:

Sir—An article under this title appearing in your issue of February 8 will stand a little analysis. In it L. P. Hills describes a lift-pipe for Pachuca agitators which has been patented and in use for some time. It probably has all the good points that he claims for it. Possible by using the form of lift-pipe described, thickening before agitation could be avoided, but whether sufficient saving in cost of plant and operation would result to justify the installation of increased agitator capacity which would be necessitated by settling and decanting in Pachucas, is decidedly questionable. Further, Mr. Hills has not calculated the number of washes which

would be required to reduce the value of the solution in his agitators to a point where the agitator charge could be discharged to waste. Assuming the following conditions: A recovery of $10 per ton of ore, battery solution $1 per ton, milling solution ratio 6 to 1, recovery in milling 40%, thickening in agitator from 6 to 1 to 3 to 1, agitation to recover 60% remaining in the ore, thickening to 2 to 1 in agitator, and diluting to 3 to 1 with barren solution *ten times,* there would still be $0.14 dissolved gold per ton dry in the charge. Mr. Hills naïvely claims for a plant designed according to the description in his article, "simplicity, cheapness of installation, economy of power and labor, low maintenance cost, flexibility of operation." I don't question the simplicity. The article under discussion was evidently to once and for all settle the question of intermittent *v.* continuous agitation in favor of the former. Its author directs his broadside at the Pachuca agitator while ignoring flat-bottomed agitators entirely. It will not be long before Pachucas are a thing of the past, and one of the salient features of the flat-bottomed agitator it its adaptability to continuous agitation. By regulating the height of discharge, any desired concentration of coarse and quick settling particles can be secured. In any agitator containing a known tonnage of quick settling particles and having a known feed per hour of same, the hours of agitation of quick settling particles is determined by dividing tonnage by feed. By regulating height of discharge in a flat-bottomed agitator the former can be controlled, allowing time of agitation to be nicely adjusted. There remains, of course, the fact that some quick settling particles will escape untreated from the first agitator, but assuming four or more agitators and applying the law of probability and chance, it can be readily determined that all particles with the same settling rate will be to all intents and purposes equally treated. Regulation of time of agitation upon the basis of settling qualities can be readily obtained in any flat-bottomed agitator, especially the Dorr agitator, with its central life. The saving in labor of continuous over intermittent agitation cannot be questioned by anyone who has used both, and properly designed continuous agitation plants allow a number of nice adjustments impossible in using the charge system.

NOEL CUNNINGHAM.

Timmins, Ontario, February 17.

IMPROVED CONE

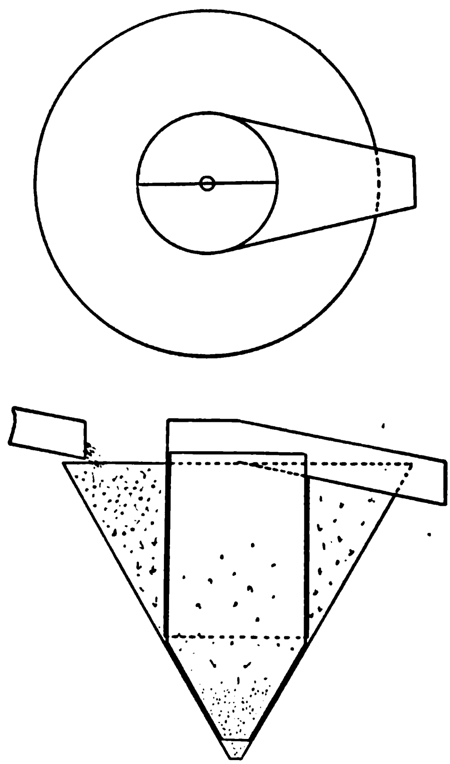
By Douglas Waterman

(April 20, 1912)

Cone classifiers are in general use in cyanide plants to separate sand from slime. The ideal efficiency is attained when the overflow is free from sand. The volume and specific gravity of the pulp determine the size of the cone. It is desirable, in order to save head room, to have the cone as small as possible. The usual form of

cone has a central cylindrical intake and peripheral overflow. Two factors operate to prevent the quiet settlement of the sand. One is the diminished sectional area where the pulp enters the cone, and the other the rapid rise of imprisoned air bubbles.

A simple device, which permits of a great reduction in the size of the cone, was introduced by W. G. Mosher at the Butters San Sebastian mine in Salvador, and has been generally adopted in the Butters plants. At San Sebastian this device has more than doubled the capacity of the cone. This improved cone is shown in the cut here-



IMPROVED CONE

with. The pulp flows directly into the cone, rises through the central cylinder, and overflows by means of a spout attached thereto. The stream enters the cone at the point of greatest area, where the force of the current is readily dissipated. The rising current in the cylinder is of low velocity and entirely free from air bubbles. The cylinder is divided by a partition extending to near the point of the cone. This prevents the vortex which would otherwise be formed by the discharge of the spigot. The annular space between the side of the cone and the cylinder is ½ to ⅝ inch.

FILTRATION

FILTER-PRESSING SLIME

By M. W. von Bernewitz

(September 17, 1910)

*The filter-press, which has done so much for the successful treatment of the sulpho-telluride ores of Kalgoorlie, Australia, has a rival in the vacuum process; and this paper is written with a view to arousing more interest in the cheaper working of the press, it being doubtful whether its efficiency in extraction can be improved.

At the Associated Northern Blocks, where these notes were compiled, the average value of the ore of late has been $6.48 per ton. It is dry crushed in three No. 5 Krupp mills. The crushed ore is roasted in six Merton ordinary type furnaces. This is ground in eight 5-ft. grinding pans with weak KCN solution. The slime from the pans is thickened, run into vats, and agitated in the usual way. The three presses are of the Dehne type, with 50 frames 39 by 39 by 3 in., and 50 plates for the filter-cloth, and the angle lever screwing-up gear. Each press holds 4.75 tons of dry slime. Filling is done by a Pearn 3-throw pump running at 20 r.p.m., with plungers 12 by 10 in., taking about 15 minutes to fill a press at 60 lb. pressure, that is, if the slime is fairly thick. In filling there appears to be a settlement of the heavier slime particles toward the bottom of the cake.

In the centre of nearly every cake of slime is a thin seam of fine sand, due to the solution, during filling, going out on either side. It is a curious thing, and possibly has some influence on subsequent washing. Nothing is gained by filling presses at high pressure. Sixty pounds has been found right for speed in filling and economy in power. A pump similar to that used for filling, but running at 13 r.p.m., is used for washing the presses. Experiment has demonstrated that 75 lb. is the right pressure for good and cheap washing. At 120 lb., 5.2 tons of wash is pumped through in a half hour, and at 75 lb., 4 tons, this giving less than one ton of wash per ton of slime. This varies with the way the press has been charged. In washing, the solution enters the high pressure plate, passes through the cloth, then through the 3-in. cake, and finally into the low-pressure plate and out.

The drop in value is fast from the start to after 10 minutes, then decreases slowly. The sump solution assayed a trace of gold and tested 0.06% KCN. During filling the vat charge was only 0.036%. With the low-grade ore treated most of the gold was dissolved in the pans and an agitation of only four hours was necessary with 0.036% KCN. This was run from the presses to the zinc-boxes, but was made up with strong solution before passing them, thus giving 0.06% sump solution, when the agitator charge was only 0.036.

Mention was made of the settlement of slime particles during filling. Careful sampling gave an average residue of 30c. per ton from the top, centre, and bottom of the cake, the sand parting being

*Abstract from Proc. Aust. Inst. Min. Eng.

the same; at the same time, it would seem that the coarser material would let more wash through to the detriment of the finer portion of the cake. This is true in very sandy slime. A water wash of 5 min. after 25 min. KCN wash is of little benefit, and tests show that it takes 20 min. for the water to thoroughly displace the cyanide. Blowing or aerating a press charge after filling and washing is of no benefit.

As to the amount of soluble gold left in the residue, samples washed for some time with water showed 20c. per ton less than that discharged. It will not pay to go on washing until the soluble gold is all out, and it is doubtful whether it could be washed out. With high-grade ore, or when the roast is poor, the difference is much greater. With a residue assaying $1.92 per ton the washed samples would be 72c., and on a 96c. residue, about 48c. per ton. This shows the faulty washing (or something else), that goes on at times in the press. Washing presses from the high or low-pressure plates gave no difference, only keeping the cloths a trifle softer. Changing all the cloths regularly has been done with fair results in better washing, but did not pay. One trouble was that the cakes would not dry properly for some days, this being bad for discharging to belts. Cloths here last from 6 to 8 weeks. Hessian was tried under the cloths for some time to save them from cutting, but the advantage was not equal to the expense. Corrugations on the plates, especially the high pressure, get full of lime compounds, which interfere with the free flow of the wash. It is fairly soft and is scraped out, but on the low-pressure plates the deposit crystallizes very hard. Assays of this have given as high as $24 per ton.

In six years at this plant only one frame was broken, but an average of two plates are broken monthly, the number of high and low-pressure plates being equal. The breakages may be due to shock from the filling pump, such as a sudden rush of pulp; or one channel in a frame may be choked and all the pressure be on one side; or from turning on the wash too suddenly, or at too high a pressure. A blocked channel is the most usual cause. Plates cost from $16.80 to $24. The Oroya-Links makes its own plates, which are good and cost $14.40 each. Martin's foundry, in South Australia, made a plate, the inside part being separate from the outside or frame part, so that there could be no breakage in the usual place. If one plunger of the filling pump is not working, the pressure rises and falls a good deal, and several breakages were due to this cause. To catch the leakage from a press 20-gauge galvanized corrugated iron is used, these lasting for two years or more. After washing, the press is dried, for two minutes, with air at 90 lb. pressure, each using 2640 cu. ft. of free air. The residue contains 22% moisture, and although this is high, it is cheaper to lose a little weak and barren solution than to use expensive air.

Two men dump 11 presses in 8 hours, being paid 64c. per press, having to keep them properly clothed, greased, and cleaned. The cycle of operations takes up the following time: filling, 15; washing, 30; drying, 2; dumping, 30; total, 77 minutes. During 1909,

44,163 tons was treated for an average extraction of 93% and costing for agitating and cyaniding 31c., and for filter-pressing 25c. per ton. The present cost of disposal of residue is 10c. per ton.

Over 60,000 tons of residue was re-treated. The system consisted in trucking to the belt-conveyor, elevating to a mixer, mixing with sump solution, and running into an agitator, although no agitation was necessary, this acting really as a storage vat; filling the press and washing 10 minutes. On the last 12 months' re-treatment of 24,675 tons the average was 78% extraction at a cost of 78c., of which re-mixing cost 44c.; pressing, 24c.; and disposition of residue, 10 cents.

Some of this was hard to get at, hence the re-mixing cost (trucking, elevating, and mixing) is high. The cyanide consumption never exceeded 3 lb. per ton of residue. The vacuum plant may be cheaper in first cost than the press plant; but its efficiency is no higher, although costs are a little lower on old dumps. Some method of emptying the filter-press without having to open it is much desired, as it would do away with some of the cost of discharging, and part of the cost of disposing of the residue, and the press would only have to be opened for the renewal of cloths. Sluicing is the only method possible, and unless cheap water can be procured, this would not pay. Experimenting on this line, one end of the charging channel was left open, and the wash valve either to the high or low-pressure plates, or both, was also left open. Pumping water for half an hour at from 75 to 120 lb. pressure, from 4 to 6 tons was used and the best test showed only one-third of the slime washed out. A great deal depends on the state of the cloths; as it seems that, as soon as the water makes a channel in many cakes it simply runs away and does no further washing. These methods are too slow and expensive in power and water.

Mr. James, recently of the Golden Links, has a small press of local design fitted with a device of local invention for sluicing and had success in sluicing out cakes of oxidized-ore slime. The frames are cast with a slot near one corner of the bottom fitted with a narrow door with rubber joint, worked by a lever and cam. A 4-in. pipe runs the length of the press with a bend and nozzle for each frame. When the press has been washed the doors are opened and the pipe lifted up by an easy method, the nozzles pointing up into each frame. Water at 50-lb. pressure is used and the pipe slowly turned through an arc of 90°, the slime coming out through the doors into a launder. About 2 of water to 1 of slime would be sufficient to wash out the porous slime in the presses of Kalgoorlie. To install this system would mean fitting or casting new frames with discharge slots. The other thing necessary is water at 36c. per 1000 gal.† Some more work on these lines would be a great

†This may need some explanation for American mining men. Under the new arrangement with the W. A. Government, the mines on the Golden Mile get water for ordinary purposes at $1.68 per 1000 gal., and for sluicing away residue, at 36c. per 1000 gal., but there is doubt whether we would be allowed to use this cheap water in the presses.

value to the cheaper working of the filter-press, and it is to be hoped that somebody will follow up this subject.

OLIVER FILTERS

(March 11, 1911)

The Editor:

Sir—In Philip Argall's article entitled 'Review of Cyanidation in 1910, published in the *Engineering & Mining Journal*, January 7, 1911, he states: "The drum form of continuous filter, of which we may take the Oliver as a type, appears to be coming into use in the United States and Mexico; they all leave much to be desired; the complication of automatic valves on vacuum, pressure air, and wash-water prove troublesome." I do not intend to discuss here the relative merits and demerits of various types of filters, but wish to take exception to statements by an engineer of such high standing as Mr. Argall that are evidently based, not on the observation of the operation of an Oliver filter, which has never had "valve troubles," but upon the difficulties experienced by a rank imitation of the Oliver filter which has lately been built in Denver and which filter has necessarily had to omit the vital features incorporated in the Oliver. While I do not object to having the Oliver referred to as a type, I do most strenuously object to having the filter blamed for the deficiencies of inferior imitations. In substantiation of the fact that the Oliver does not have "valve troubles," I beg to state that in the four year that I have been building these filters, no changes have been made in mechanical principle or general design of the valve.

EDWIN LETTS OLIVER.

San Francisco, February 25.

RECENT PROGRESS IN SLIME-FILTRATION DEVELOPMENT

By C. S. Haley

(July 1, 1911)

The threatened revolution of the cyanide process from a chemical standpoint by the electrical regeneration of solutions has of late provoked a good deal of discussion, and justly. However, the revolution in the mechanical handling of pulp, solutions, and precipitate in the course of the last few years has perhaps not received that share of the attention of the metallurgical press which is its due; yet it has been no less far-reaching in its effects, although more gradual in its development. Of the different phases of this revolution, perhaps the most marked has been that of the change and development in the methods of handling and treating the more flocculent and troublesome portion of the crushed ore classified as slime. It has been my privilege to operate or assist at the installa-

tion of practically all the leading types of slime-filters at present on the market; and the results of an experience extending over ten years, not only in connection with the treatment of gold and silver ores, but also with the same type of work in manufacturing processes, has convinced me that there is still much to be done in connection with this line of work.

Ten years ago the present common types of vacuum-filters were practically unknown in the mining world. The necessity for a treatment of the more flocculent pulp different from the ordinary leaching methods applicable to the heavier sand, was early realized; but the treatment as first applied was cumbrous and unwieldly. The problem of securing agitation presented its difficulties, if the maximum extraction was to be obtained in the minimum of time. However, having expended so much time, power, and chemicals, on the solution of the gold, thereby increasing the amount of capital tied up in the mixed pulp and solution, a greater problem arose in the separation of the dissolved metal from the worthless pulp. It is this problem that I wish to discuss.

The first, and simplest, attempt at the solution of this problem was the ordinary decantation method. After agitation the slime was allowed to settle for as great a length of time as economy permitted, and the clear solution was drained off by decantation pipes or siphons. Various modifications of this process were made, and are still used today in more remote localities; but several grave difficulties arose. The first was the loss of time—a considerable factor. Next came the difficulties introduced in the refining of the product by the almost unavoidable introduction of a certain amount of fine silica into the zinc-boxes. Last, and most important, came the difficulty of giving a good wash to the pulp and the necessity of discharging good gold solution with the waste of pulp at the end of the operation. After spending money to get gold into solution, any good engineer regards it as little short of criminal to deliberately throw any portion of that solution away. The decantation process, however, reigned supreme in this country for a considerable time. In Australia the method of separation by pressure was long ago in use; but the cumbrous methods required to clean the presses after filtration were a great drawback. The final introduction of this method into this country was signalized by a remarkably ingenious invention which did away with this objection very completely. This will be taken up later more at length.

In Colorado the introduction of filter-leaves into the slime-tanks was developed into a system of gravity filtration which had its drawbacks. Chief of these was the difficulty of giving a proper wash. A downward-percolating solution will not channel in ordinary sands; but in the case of slime it will not do anything else; and channeling is of course fatal to thorough washing. Further, the time element enters in to too great an extent. However, an ingenious mind conceived the idea of applying a vacuum-pump to these leaves and drawing the gold solution through them, leaving the pulp caked on the outside. Washing was accomplished by

lifting the leaves, cake and all, bodily out of the original tank, and into another tank containing barren solution, which, on being drawn through the cake, displaced the gold solution and carried it to the precipitation boxes. A variation of this process, by means of which all the operations were carried on in the one tank, by the introduction of different solutions, brought with it improved vacuum machinery, and was soon made, and, as well as the older process, is now in wide use.

As this process is in general use, it may be well to give a brief résumé of its salient points. The pulp is introduced from the agitator, or stock tank, in a semi-liquid condition. Before the slime has a chance to settle in the filter-tank, a vacuum is applied on the filter-leaves, and gradually increased, its limits depending on the character of the ore and the method of treatment. A cake, varying in thickness from three-quarters of an inch to an inch and a half, is formed on the leaves. The surplus pulp is then returned to the stock tank, and the wash-water introduced. This wash-water is drawn through the cake, displacing the greater part of the dissolved metal which remains in the cake. The number of washes applied is of course also dependent upon local conditions. After the surplus wash-water is drawn off, the cake is discharged by the distending of the leaves from the inside by the introduction of water. The main difficulties left to be overcome by this method were still the time element, and, with the more difficult types of ores, the replacement of dissolved metals. While this type of filter is a vast improvement in both of these respects over its predecessors, it is still imperfect. Furthermore, the clogging of the filter pores by fine silica and lime renders frequent acid washing necessary, and renewal of the filter-leaves imperative. The presence of a chain-block for the repair of the slightest leak is imperative, and time is lost in the discovery and repair of leaks, especially as the filter grows older in use. About four or five years ago a filter was put on the market from Colorado which attempted to wash a mixture of sand and slime by upward percolation. This is merely mentioned to show the vanity of human endeavor.

The introduction into this country of the process of pressure filtration was greatly facilitated by the addition of the sluicing-bar, which did away with the principal objection to the filter-press, in that the cleaning of the press-chambers was automatic and speedy, though in dry countries the amount of water necessary for sluicing may be important. Pressure-filtration gained its greatest favor from the fact that, with its use, agitation, with all its use of power, is not necessary. Instead of bringing the particles of ore into contact with the solution, the solution was forced into close contact with the particles of pulp by means of pressure. The heavy and cumbrous presses, by reason of their cost and the difficulty of transportation, have been one great drawback to the use of this process in regions which are at all inaccessible. Its success where conditions admit of its use has been, on the whole, admirable.

Contemporaneously with the development of this form, a rather ingenious pressure-filter which is in no sense a filter-press, was a good thing which came out of Utah. It consists of a large cylinder resembling in outward appearance a compressed-air receiver mounted at an angle from the horizontal. Mounted on a track inside this cylinder is a set of filter-leaves running longitudinally and decreasing in height, of course, from the centre out. Under pressure of about thirty pounds the solution was forced through these leaves, and came out through nipples in the grate to which the filter-leaves were attached. The cake was rapidly formed on the leaves, and the excess pulp returned to the stock tank when a definite time had elapsed. Wash-water was then introduced, and run through the cake for a definite time. Pressure was held on the cake by means of air when the wash-water was returned, and it was thoroughly aerated, producing a very dry cake. When the cycle was completed (ordinarily in about an hour) the end-gate was opened, and the filter carriage let out on a track by gravity, which overhung the discharge chute. The idea was to discharge the cake by internal pressure of water, or steam, where available, but the cake formed is generally so dry that the operator loses much time in scraping and hosing it off. This filter has the advantages of giving an excellent wash and being simple in its operation; but weighed against this is the necessity of having a man constantly watching it and frequently needing a helper to discharge or re-seal the filter.

This amount of labor necessary in the operation of a filter, as well as the time lost in discharging the cakes, has resulted in many attempts to develop a continuous filter; one which would run by itself and be constantly discharging the pulp. Many solutions of this phase of the problem have been offered in the last few years, with varying success. One type which is coming into wide use on account of its convenience and practicability has recently been developed. This filter consists of a revolving drum, whose spokes are vacuum pipes or compressed-air pipes, as occasion demands. This drum is immersed in the pulp nearly to its axle, and a gentle vacuum causes a comparatively thin cake to collect on the outside of the drum, whose canvas surface is protected by wire. The pulp is slightly agitated in the filter-tank by means of compressed air, and as the cake forms on the surface of the drum it is lifted out of the tank by the revolution of the latter. As it rises and begins to harden, it passes under a dripping spray of wash-water, which is sucked through the drum surface and washes the thin cake. The cake is carried around until it comes within about three feet of a scraper, when, by an ingenious valve arrangement, the suction in that part of the drum is changed to a pressure. This lossens the cake, the wire protects the filter-cloth from the scraper, and before re-submergence that section of the drum is thoroughly cleaned and ready to take another charge. This filter has lately been imitated in Colorado. The valve action in the California patent is simple; the main difficulty arises in case of stoppage of the filter for any

cause, when it is a trifle hard to start. The problem of washing seems to be well handled in this filter, the time element no longer enters, and the filter is automatic in its operation. The repair of leaks is accomplished by slipping a small piece of canvas under the wire. One difficulty, however, remains. In the case of a heavy sliming ore, such as a stephanite type, once the pulp is in a cake form it is next to impossible to give it a perfect wash, as a certain amount of channeling is bound to occur. If the cake should be broken up into its various particles of slime before being submitted to a wash, washed by compressed air, and then gathered together again, a practically perfect wash could be obtained. However, in the different filters that have attempted to handle this problem (most of them of the continuous type) the increased mechanical cost and difficulty have far outweighed the saving effected.

On the whole, the development of slime filtration during the past few years has merely exemplified the extreme quickness of metallurgists in adapting themselves to new conditions imposed upon them by the adoption of new processes of treatment.

SLIME FILTRATION

By George J. Young

(October 28, 1911)

*Much has been written concerning an accurate definition of the term 'slime,' but no comprehensive definition seems to be generally accepted. An excellent study of the nature of slime, by R. E. Ashley, was published in the *Mining and Scientific Press* of June 12, 1909. The accepted present practice in milling work is that all material in a pulp finer than a 200-mesh screen is considered as slime. The definition that a slime is the unleachable portion of a mill-pulp is also in use. A more comprehensive definition than the foregoing is that a slime consists of a mixture of sand finer than 150 or 200-mesh screen with an amorphous clay-like material, consisting principally of hydrated aluminum silicate.

The general method of slime-treatment is to agitate the slime with a cyanide solution for a sufficient time to dissolve the gold, and then, either to filter off the surplus solution and displace the remainder with water, or to thicken the slime by settlement and decantation, and then to filter and displace the remaining solution by water. The mechanical appliances in use for filtration are grouped as follows:

These filters are continuous in action.

I. Suction-filters, or filters in which a vacuum is used to accelerate filtration.

A. Appliances using a thin slime-cake and practically continuous in this action. (Oliver and Ridgway filters.)

*Abstract of a paper presented at the San Francisco Meeting of the American Institute of Mining Engineers.

B. Appliances using a thick slime-cake and practically continuous in tion. (Moore and Butters filters.)

II. Pressure-filters, or filters in which hydrostatic head, compressed air, or pumps are used in order to secure greater pressures than are possible with a vacuum-pump.

These filters are intermittent in their action.

C. Ordinary filter-presses.

D. Sluicing filter-presses (Merrill filter-press).

E. Filtering-chambers or cylinders; filters in which the filtering-basket is enclosed in a cylinder. (Burt, Kelley, and Swetland filter-presses.)

III. Centrifugal filters, or filters in which centrifugal force is used to separate solution from slime.

The filters in sections I and II with the exception of the Ridgeway, employ vertical filtering-surfaces. The Oliver makes use of a revolving cylindrical surface as a filtering surtace. Centrifugal filters are in process of development and have not as yet secured any foothold in gold and silver metallurgy. It is not improbable, however, that some comparatively simple filter based on the use of centrifugal force will be perfected, and will successfully compete with the other forms. At present the suction filters are in greatest use. Of the pressure-filters, the ordinary filter-presses have gone out of use, except as clarifying presses, and filters of groups *D* and *E* only are in use.

The development of slime-filtration is of interest. Filter-presses and filtering-beds in vats were first used. The filtering-beds were soon discarded and the filter-press systematically developed. The size of the press was increased, mechanical devices to facilitate discharge and decrease the proportion of labor required were invented and introduced; but in spite of all this the cost of treatment in filter-presses remained high. In western America the filter-press never received much recognition, but in Australia filter-pressing was extensively introduced, and slime was successfully handled by this method. It remained for an American, Charles W. Merrill, to complete the last improvement in the filter-press. By the introduction of the sluicing system the slime-cakes can be washed out of the filter-cells and the press operated without opening or separating the filter-plates for each charge. This improvement reduced the labor and cost, and increased the effectiveness of the filter-press. The Merrill press represents the culminating point in the filter-press line of development in slime filtration.

The Moore filter was the first suction-filter in the field, and introduced the idea of the canvas-covered filtering-cell immersed in the slime-pulp and utilizing suction to draw the solution through the walls of the cell and to build up a cake. The necessary transfers are made by lifting the filtering-basket out of the pulp. The Butters filter introduced the idea of a stationary filtering cell, and

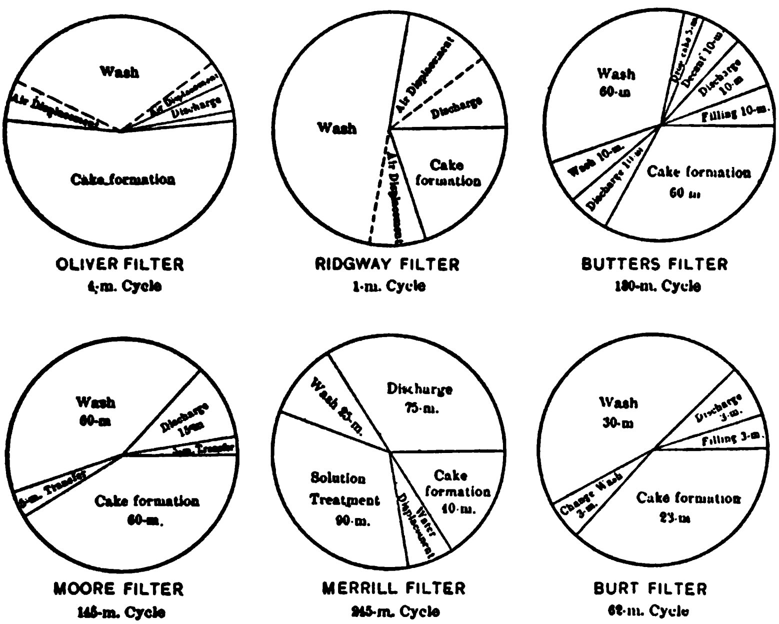
effected the transfers by pumping the slime-pulp and wash-water from the vat in which the filtering-cells were immersed. The relative merits of the two systems have been sufficiently discussed in the technical literature. Both the Moore and the Butters filters have reached a point where little or no further improvement seems possible. Like the Merrill, either one of these systems will satisfactorily meet the requirements of slime-filtration. The combination of the ideas involved in the filter-press and the suction filter is seen in group *E*, or the filtering-chambers. The Kelly, the Burt, and the Sweetland may be compared to a Butters filter installed in a pressure-tank.

The effort to secure a continuously-acting filter has resulted in two important types being developed, of which the Ridgway and the Oliver are the best known. Both of these filters utilize a comparatively thin slime-cake. Both operate very successfully, and compared with the thick-cake machines have decided advantages, brifly stated as simplicity of design; probably lower capitalization-charges for equal capacities; lower operating costs; and less attention required in the operation.

With the exception of the Oliver filter, the general method of operation of both suction and pressure-filters is the same. The slime-pulp is delivered to the filter in the proportion of one of dry slime to from three to one of solution. The pulp is forced into the cells of the pressure-filters and a cake formed against the canvas walls of the cells, the surplus pulp, if any, is withdrawn, and wash-water forced in until the contained solutions are displaced. The cake is then forced off from the canvas surface, either by water or air, or a combination of both, and sluiced out. In the vacuum-filter the filtering-cells are immersed in the pulp, a vacuum is formed, and a cake built up; the surplus pulp is then withdrawn either by lifting the filtering-cells out or by withdrawing the pulp by pumps, and the cakes are immersed in water for washing. In the Moore filter the cakes are discharged by forcing them off from the cell by water or air and dropping into a hopper for sulicing away; in the Butters the cake is forced off in the same way, but while still immersed in the wash-solution. The wash-solution is then withdrawn, either by decanting or pumping, and the slime-cake and surplus wash sluiced out. The Oliver filter performs the operations of cake-formation, washing, and discharge in continuous sequence. Three steps may be designated as common to all these filters: cake-formation, washing, and discharge. The cycle of operation of the more common forms of filters is shown in the accompanying figure. Typical examples have been taken in each case.

The condition under which slime-cakes are formed and washed are the critical points to be considered; the discharge and sluicing away of the cake is a comparatively simple matter and requires no special comment. My experimental work was largely confined to suction-filtration, and pressure-filtration was only briefly studied. Experiment on the variation of the filtering rate with variable thickness of slime-cake both while building up and in clear water, indi-

cate that the filtering rate during building up a cake is greater in the pulp then in clear water for thin cakes, while for the thicker cakes the reverse is true. The general effect of increase of pressure upon the filtering rates of cakes of varying thickness is to increase the filtering rate. This is more marked with the thin cakes, while with thick cakes the effect of an increase of pressure is to increase the density of the cake and thus reduce its permeability. With higher pressures this effect is more marked, and indicates that a point would soon be reached where the increased pressure would result in decreased filtering rate. This is particularly true of slime



CYCLE OF OPERATION OF VARIOUS TYPES OF FILTERS

containing a small proportion of sand, and much less so with slime containing a large proportion of sand. In the use of both the Moore and Butters systems, experiments should be made with different intensities of vacuum, for it may be found that a vacuum lower than the maximum obtainable with the available apparatus will give a higher filtration rate, and thus decrease the time for both building up and washing. Many experiments were made upon the effect of temperature on filtration, the character of the slime-cake, the effect of the character of the filtering surface and the method of its support, the rate of building up the cakes, and washing the cake. For details of these experiments and the curves showing the results attained, reference should be made to the original paper.

Data were secured from a number of slime-plants in Nevada through the courtesy of the different managers. The table shows the results of a number of physical analyses of slimes obtained from these plants. For purposes of comparison the type slime, the sand-slime mixture, the clay slime, and the filtering rates are included. The lower table gives the data of the slime-plants from which the mill-slimes in the upper table were taken.

PHYSICAL ANALYSES.

	Type.	Sand-Slime. 50 Per Cent. of Sand.	Fire-Clay Clay-Slime.	A.	B.	C.	D.	E.	F.	Size.
				Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	m.m.
Less than 2 min. settlement. + 100 screen	0.10	0.43	22.3	4.5	1.70	2.3	6.2	2.3	6.6	0.17 — 0.22
— 100 + 150	1.30	3.65	14.0	6.9	9.0	12.30	11.3	4.46	6.8	0.11 — 0.17
— 150 + 200	1.20	6.60	4.6	7.3	9.0	12.0	8.35	7.0	10.7	0.05 — 0.11
— 200	26.0	13.80	3.3	30.2	43.70	34.0	38.05	49.54	68.5	0.01 — 0.05
More than 2 min., less than 4 min.,	8.5	6.3	2.0	4.25	4.90	11.0	5.5	3.6	0.7	0.005 — 0.02
More than 4 min., less than 8 min.,	11.6	8.4	2.2	3.6	4.30	6.7	6.2	3.8	0.65	0.005 — 0.01
More than 8 min.,	54.3	30.8	51.6	34.25	27.10	22.7	24.4	30.3	7.05	0.0 — 0.005
Thickness of cake, inches,	1.00	1.00	1.00	1.00	0.75—1.0	1.25—175	1.25	1.75	1.5—2	
Filtering-rate Building.	5 lbs.	10	3	11.6	26	8.3	7.25	30		
Filtering-rate Washing.				10.0	16					
167 per 100 sq. ft.										
Sp. gr. dry slime,	2.62	2.67	2.57	2.58	2.62	2.62	......	2.684		

CHEMICAL COMPOSITION.

	Type.	Sand-Slime.	Fire-Clay.	A.	B.	C.	D.	E.
	Per Cent.		Per Cent.	Per Cent.	Per Cent.	Per Cent.		Per Cent.
Silica,	74	......	68	74	77.2	68	......	84.4
Alumina,	10.9	......	22.9	12.8	13.9	15.8	......	13.5
Ferric oxide,	1.1	......	1.1	1.4	1.9	2.0	......	2.9
Lime,	1.0	......	0.0	0.8	1.0	1.0	......	5.2
Magnesia,	0.6	......	tr.	0.0	0.5	0.0	......	1.2
Aluminum silicate (Al_2O_3 2 SiO_2), calculated,	22.6	......	49.9	27.9	30.8	34.4	......	29.43

DETAILS OF FILTER PLANTS

	A.	B.	C.	D.	E.	F.	G.
Type of filter,	Butters.	Butters.	Butters.	Butters.	Merrill.	Kelley.	Butters.
Number of units,	2	2	2	2	3	1	1
Number of leaves per unit,	60	95	168	72	64	10	100
Number of leaves,	120	190	336	100	192	.....	100
Area of leaf, sq. ft.,	100	91	100	92.6	41.4	....	100
Total filtering-area,	12,000	17,290	33,600	9.262	7,980	360	10,000
Tons slime per 24 hr.,	175	250	1,000	150	216	100	150
Tons slime per 100 sq. ft. per 24 hr.,	1.4	1.44	2.97	1.62	2.72	27.7	1.5
Slime-pulp consistency, water : slime,	3:1	3:1	1.5:1	2:1	2:1	1.1	2:1
Filter-support,	slats	slats	matting	slats	plate	wire netting	slats
Thickness of cake, in.,	1	0.75-1	1.25-1.75	1.25	1.75	1.5-2	1.5
Moisture in cake, p.c.,	38	35	35	33	......	12	29
Time forming, hr.,	1	0.5	1-1.33	1	0.66	0.08	2
Time washing, hr.,	1	1.0	1.41-1.66	1.25	0.41	0.20	2
Time transfers, hr.,	0.75	0.83	0.59-0.51	0.5	3.01	......	1.88
Total time-cycle, hr.,	2.75	2.33	3-3.5	2.75	4.08	0.5-0.75	5.88
Filtering-rate per 100 sq. ft. per min.,	11.6	26	8.3	6.83	30	......	......
Filtering-rate per min. for wash,	10	16.3	......	......	......	......	......
Tons of solution and wash per 24 hr.,	848	1.356	2,000	350-400	328 340	75	484
Canvas used, oz.,	12	12	12	12	18	12	10
Life of canvas, months,	8	6	41 est.	18	12	0.5-1.5	......
Frequency of acid-wash, days,	20	21	30	21	60	15	30
Approximate cost per ton slime,	$0.18	$0.27	$0.075	$0.25	$0.288	$0.35	$0.119

The practical conclusions which may be drawn from the experimental work may be summarized as follows: 1. The proportion of clayey material in ores which are to be subjected to 'all-sliming' and filtration should be maintained at a minimum. 2. The slime-pulp should be as free as possible from sand coarser than 150-mesh, and as large a proportion of the pulp as possible should consist of material passing 200-mesh. 3. The slime-pulp before filtration should be settled as thick as possible consistent with ready handling by pumps and in pipes. 4. The temperature of the slime-pulp should be maintained between 20 and 30°C., or higher. 5. The temperature of the wash-water and the pulp should be the same. 6. Vacuum-pressures should be varied until the proper intensity for the given slime is obtained. 7. Where the very clayey slime is to be filtered, as much fine sand should be crowded into the pulp as it will carry without undue settling and clogging. 8. No. 10 canvas supported by slats gives the best all-round service for the thick cake, and No. 12 canvas on wire netting answers the requirements for the thin-cake filtering machines. 9. With slime containing a large proportion of colloid or clayey material pressures greater than those obtainable with vacuum apparatus are of questionable advantage. 10. With slime containing a large proportion of clayey material the vacuum-filters should be used. 11. With slime containing a small proportion of clayey material and much fine sand both vacuum-filters and pressure-filters could be used with perhaps equally good results. 12. With slime containing much coarse and fine sand the chamber-filters with air-agitation and high pressure would perhaps give the best results. 13. Of the vacuum-filters, the thin-cake continuous filters are a decided improvement over the thick-cake filters.

OPERATING A STATIONARY FILTER

By H. G. Smith

(October 19, 1912)

At the Concheño mill of the Concheño Mining Co. a standard 30-leaf semi-gravity type Butters filter is in use. The operation of this required a longer solution wash than the crushing plant permitted and trouble was occasioned by the cracking of the cakes during the transfer of charges, which made an even wash impossible. In the ordinary cycle of operations 15 minutes was consumed in pumping the excess charge, and about the same time was required to fill the filter-box again; thus the cakes were exposed to the air about 30 minutes, during which time large cracks were formed, and filtration practically came to a standstill.

To prevent the cracking of the cakes and at the same time to devote the 30 minutes time to useful account, I hit upon the following expedient. Instead of pumping back the excess charge of slime, and filling the filter with wash solution from the bottom of the tank, I connected the solution-line with a 4-in. pipe and

valve, and to it a 4-in. pipe, perforated with ¾-in. holes with 2-in. centres, which rested on the filter leaves and extended the length of the box. Then, after the formation of cake on starting the pump, I ran in the solution wash through the perforated pipe at the top of the charge, keeping the leaves covered all the time. In this way the 30 minutes consumed in filling and emptying the filter-box is utilized in cake-making and washing, and the leaves being submerged the cake has no chance to crack, an even wash is assured, and an additional 30 minutes wash is possible, or the cycle can be shortened to that extent.

Of course there is some mixture of slime and solution but not as much as might be supposed. By watching the discharge of the pump the change can be detected almost instantly when the slime charge ends and the mixture of charge and solution comes in. Then the mixed part is turned to another tank, and pumping continued until the solution runs clear; when the pump is stopped until time to change the charge. The mixed slime and solution is allowed to settle, and the thickened slime is run to the agitators again, while the clear solution is turned to the mill solution or to the zinc-boxes. Possibly more solution is precipitated than would otherwise be the case, though not necessarily, as the dilute pulp can be run to the settlers, or thickeners, and the overflow of solution can be handled by passing to the zinc-boxes or run into the circulating mill-solution. More solution is handled and more power is consumed than ordinarily, but in any case there is always a certain amount of mixed slime and solution to be taken care of under any conditions.

In some cases it might be of advantage to carry the method a step further and replace the solution wash by the water wash; this could be done by having the sump tanks graduated so a definite amount of solution could be pumped and the stock of solution could be kept constant. This might be of advantage where the solution wash is very weak (as in some gold plants) where the mixed portion (which would have to be run to waste) could be kept at a low cyanide content. I offer this method to my co-workers for what it is worth; and while the application is original with me, I make no claim other than that by using the system the cracking of the cake is avoided (assuring an even wash) and that the time saved in the cycle over the ordinary method more than offsets the objection of the extra pumping solutions. Where a saving in the cycle of operations of 30 minutes can be made another charge can be run through the filter in the course of 24 hours time, which is quite an object where the filter is crowded to keep up with the crushing departmeut.

(January 18, 1913)

The Editor:

Sir—I was interested to read in your October 19 issue of the success attending the introduction of the method of simultaneously

replacing the pulp with wash solution in the working of the stationary type of vacuum-filter at the Concheño mill, in Mexico. H. G. Smith, who describes the innovation, takes undue credit to himself as the originator of the idea. *The Engineering & Mining Journal* of May 15, 1909, contains an article of mine explaining the system and its advantages, and outlining a scheme for continuous operation. For some time previous to this I had worked the method with considerable success in northern Mexico. In the concluding paragraph of his article, Mr. Smith presents the idea to his co-workers. This indicates a commendable spirit of generosity, but I would like to add that the improvement in question is protected in Mexico by Patent No. 8814, which was granted me on February 3, 1909.

A. W. ALLEN.

Lonely Reef Mine, Rhodesia, December 3, 1912.

THE MOORE-BUTTERS DECISION

(November 23, 1912)

In an appeal from the Circuit Court of the United States for the district of New Jersey, before George Gray, Joseph Buffiington, and J. B. McPherson, circuit judges, in the United States Circuit Court of Appeals, for the Third circuit, October term, Joseph Buffington, circuit judge, handed down the following opinion in the case of the Moore Filter Co. *v.* the Tonopah-Belmont Development Company.

In the court below, The Moore Filter Co., the owner of patent No. 764,486, granted July 5, 1904, to George Moore for a filtering process, filed a bill charging the Tonopah-Belmont Development Co. with infringement thereof. On final hearing, that court, in pursuance of an opinion, reported in 195 Fed. Rep. 530, dismissed the bill on the ground that infringement was not shown. Thereupon the complainant took this appeal.

As applied in the present case, the patent concerns the process of filtering metal-bearing slimes and is known as the Moore process. The respondent's filter is for the filtration of like slimes and embodies the Butters process. Both processes utilize the cyanide ore treatment and the question before us is two-fold: first, does Moore's process involve invention; and, second, does the respondent's Butters filter make use of the Moore process. The cyanide ore process came into use about 1887 and is the real foundation of the tremendous increase of gold production in the last two decades. It is now the prevalent method of treatment. In it the ore is first crushed and then placed in tanks containing a solution of cyanide of potassium. This solution percolates through the crushed pulverized mass, and being a solvent of gold carries off such gold as is subjected to its action. This is called leaching, and any crushed ore through which percolation took place was termed leachable. For example, if the ore treated was of such character

that when crushed it was reduced merely to the condition of sand, then the recovery of its metal by the cyanide solution might be effected by two methods. In the first method the cyanide solution would be poured on a bed of sandy crushed ore and be allowed to percolate through it. In its passage the solution dissolved the metal and passed off as a clear liquid to zinc-boxes, or other well-known means of reclaiming metals in solution. This very simple method was called leaching. The second was decantation, wherein the crushed sandy material after having been agitated in the cyanide solution was permitted to settle, so that the clear liquid containing the dissolved metal might be decanted. Thus so long as the crushed grain was so sand-like as to permit leaching, or would settle quickly and completely enough to permit decantation, reasonably satisfactory results were reached by the cyanide process with rich ores, but even with these the treated ore thrown on the dumps often contained large in the aggregate, though small per ton, unleached metals. This was due to the fact that the solvent had not and could not penetrate the coarse ground particles. If, however, the ore was crushed finer to permit the more intimate action of the solution a pasty mass called 'slime' was formed which was unleachable. The result of this was that great quantities of treated ore went to the dump-heap, and while laboratory filtration methods showed the presence and indeed the extraction of such metals, yet no one devised any commercial means or process by which this metal-laden dumpage or slime would be avoided or utilized. As a value-containing but unavailing feature these dumps occupied a relation to gold and silver mines like that of a slag pile to a blast-furnace or a culm bank to an anthracite mine. The proofs show the acute recognition of this grevious waste and the vain efforts of a great industry to avoid it. Thus, in *The Engineering and Mining Journal*, under date of October 8, 1892, in an article on 'The Cyanide Process in South Africa,' by Charles Butters and another, it is said:

"Another difficulty frequently encountered in the application of the cyanide process in the treatment of 'battery slime,' *i. e.*, the very finely divided material produced during the crushing, and which has a tendency to accumulate in pasty masses. These either resist the penetrating action of the cyanide or retain the dissolved gold during the leaching operation. No satisfactory method of breaking such material has yet been devised—the evil may be lessened by mixing the slimy tailing with clean coarse sand."

An editorial in the same journal, dated April 15, 1893, says:

"After a certain amount of experience with any process, its weak points are seen and opportunities for improvements present themselves. To this rule the cyanide process is no exception. One of the great difficulties experienced in this process, or indeed in any lixiviation process, is the treatment of the slimes of an ore otherwise well suited to reduction by the method. They pack upon the filter, forming beds impermeable to the solution, and even if mixed with large quantities of coarser material are rarely attacked, *al-*

though laboratory experiments will show that their precious-metal contents are extremely soluble. Of such material the Robinson Gold Mining Co., of South Africa, operating one of the largest cyanide plants on the Transvaal, has accumulated 60,000 tons, and the management has long despaired of treating it successfully, as the gold would not amalgamate nor would the cyanide permeate the mass if it were charged into vats. The average assay value was between $7 and $8 per ton, but the fineness, it is estimated, is such that it would pass a 225-mesh screen."

The same journal on August 11, 1894, contains an article on 'The Cyanide Process in the Transvaal Mines,' which says:

"One of the great bugbears of the cyanide men on the Witwatersrand has been the treatment of slime, by which is meant the very fine, or in the case of free-milling ores the clayey portions of the tailing. Many suggestions have been made for the treatment of these, but the only really practical scheme so far appears to be to allow them to dry thoroughly and by screening or otherwise to reduce them to a fine powder. This powder is thoroughly mixed with sand tailing, and the mixtures will usually percolate fairly well."

In an editorial in the same journal, in speaking of the cyanide process, it is said:

"Undoubtedly the process is well adapted to certain ores, but these appear to exist in but few localities and we have yet to learn how to extend the use of the process to more common material."

In an article on that process contributed by Virgoe to the same journal in 1894, he says:

"Filter presses have been tried in South Africa, but without satisfactory results, owing to their cost and the power required to work them. No mechanical means have yet been devised for the satisfactory separation of pulp and solution in the case of poor leaching ores. Such an invention would revolutionize the metallurgical world as far as the wet reduction of ore is concerned."

Commenting on this article a correspondent in September, 1894, wrote the *Journal*:

"Regarding the leaching qualities of the ore or tailing to be treated, I am quite in accord with Mr. Virgoe, for badly percolating material (such as battery slime) is quite the greatest bugbear of the cyanide man."

The following year (1895) Charles Butters, writing to the *Journal*, said:

"The treatment of slime is a question of importance, as at present there are many hundreds of thousands of tons of unleachable material lying useless on the hands of the various companies on the Witwatersrand."

And not only was the problem recognized and the need felt, but the agitation of it continued for years. In 1898, the same

journal, after discussing the various efforts in the Transvaal to treat rejected slime, says:

"Speaking generally, about 75% of the tailing from the Witwatersrand mills has been treated by cyanide in the usual practice, leaving about 25% to go into the slime-pit. There is therefore a large accumulation of this slime, besides that which comes from current working. What proportion of the old heaps can be treated at a profit is yet to be ascertained; but it seems possible that an appreciable addition to the gold output may come from this source hereafter."

In the same year, referring to the Australian mines, the *Journal* says:

"A great number of experiments are at present being conducted on the Kalgoorlie ores. Nearly all known processes and several never before heard of, have their advocates. Of course some valuable knowledge will be gained by all this experimenting, and just as surely a great deal of very expensive machinery will in a short time be consigned to the scrap pile. * * *

"As yet the finer slime has not been successfully treated on a large scale, but some of the ingenious adaptions of the agitation or filter-press processes, now in the experimental stage, will undoubtedly solve the problem."

Indeed the whole matter was summed up four years later when, in an article in the *Journal* of July, 1902, on 'A New Treatment of the Slime Problem in Cyaniding Talcose Ores,' a writer, Stackpole, says:

"Any metallurgist can appreciate the obstinacy of these sticky masses of mud, which, no matter how treated, would take almost a prohibitive length of time to percolate. Although experiments show that over 90% of the values in the clay is soluble, the ordinary methods only permit an extraction of 50 per cent."

The first suggestion for the solution of this world-wide problem is found in the *Journal* of December 5, 1903, being a communication from George Moore, wherein he described the process for which the patent in suit was issued to him the year following.

We deem a quotation from that article proper at this time not only as being in the line of historical sequence of the art, but for the further reason that our conclusion that Moore fully realized the scope of his invention and disclosed the same in his specification, is fortified by the fact that he had explained and disclosed it fully to the engineering world a year before his patent issued. In his article, after describing his process as installed at a certain mine and asserting that the advantages of his process were: "First, a saving of from 40 to 60c. per ton in labor; second, a saving of a like amount in extraction; third, a saving of over 50% in the cost of installation," he says:

"The saving on extraction is due to the fact that, while *the filter is in the slime-tank, and the suction in operation,* an equalizing

action is taking place, rendering all parts of the cake of equal resistance to the flow of solution and water-wash, so that, when placed in the washing tanks, a perfect displacement of solutions is accomplished. For example, we might consider that it would be possible for one spot on the 2880 sq. ft .of slime cake to have more of the coarser slime or fine sand than the other parts; then there would be less resistance to the flow at this point; therefore, the flow would be accelerated here, the slime would be brought up, and would cover this point more rapidly than the other parts until, by this increased coating, the resistance to the flow would become the same as at all other points. Thus, when lifted out from the slime compartment, the entire basket of filter is in condition for washing, and in practice, we extract all of the soluble gold."

After a careful study of Moore's patent, we have reached the conclusion that his process is a radical departure from the whole prior art and was an original and pioneer step in metal recovery by filtration. Like all important inventions its merit is its simplicity, and its novelty consists in his utilizing the simple elemental processes of nature. These processes he has, of course, neither discovered nor invented, but he has utilized them in combination in a manner never before used and has thereby secured a new result.

Briefly stated, the gist of Moore's instruction to the mining world was to filter an unleachable mass in such a manner as to cake the pasty slime so uniformly and evenly that the resistance of the slimy mass to percolation became uniform and even in every part of the cake. The result of such uniform and even cake-resistance was that when the cake was attacked by the percolating solvent its percolation was correspondingly uniform, and diffusive through the uniformly resisting cake. The solvent having by this uniform and even resistance been itself uniformly and evenly percolated through the mass, it followed that when this percolated solvent was in turn subjected to a propulsive washing current, that such propulsive current, finding no path of least resistance in the uniformly-resisting cake or the uniformly-percolated solvent, moved forward uniformly and evenly to expel such uniformly distributed percolated solvent. The creation of this uniform and even resistance in the cake is the gist of Moore's process, and such uniformity, as we shall see, is secured by the slime being submerged when subjected to suction. The crux of Moore's process cannot be better or more tersely summarized than in his own words quoted above:

"The saving on extraction is due to the fact that while the filter is in the slime-tank, and that means submergence, and the suction in operation, an equalizing action is taking place, rendering all parts of the cake of equal resistance to the flow of solution and wash-water, so that, when placed in the washing tanks, a perfect displacement of solutions is accomplished. For example, we might consider that it would be possible for one spot on the 2880 sq. ft. of slime cake to have more of the coarser slime or fine sand, than

the other parts; then there would be less resistance to the flow at this point; therefore the flow would be accelerated here, the slime would be brought up, and would cover this point more rapidly than the other parts until, by this increased coating, the resistance to the flow would become the same as at all other points. Thus, when lifted out of the slime compartment, the entire basket of filters is in condition for washing, and, in practice, we extract all of the soluble gold."

And this is only stating in other words what he set forth in his specifications. Referring to the original drawings, the specification states:

"In carrying out the process with the means disclosed in these drawings adapted particularly for use in connection with the slimes of precious metals, I introduce a solution to be filtered into a suitable tank, 1, in which I place a filtering device or means, 2, in the present instance made up of a series of filter-plates communicating with a common discharge tube, 3. A flexible or other suitable tube, 4, connects the tube 3 with any preferred form of hydraulic pump 5, and compressed-air pump 5a, and the filtering means 2 is permitted to remain within the tank 1 until the solid matter within the liquid being filtered has coated the walls of the filtering device to the desired thickness, say, for example, about 3/4 in. or more in most cases, but varying somewhat with the character of the slime which is being handled; and then the same is lifted as by pulley-and-cable mechanisms 6, out of and away from the tank, and the pump 5 stopped and pump 5a operated so as to apply air-pressure to the back of the canvas or to pass a current of air or cleansing current in an opposite direction to the movement of the liquid in the prior step, whereby the solid matter collected by the filtering device 2 will be discharged therefrom."

Applying this description to the drawings, it will be seen that the filter-leaves are completely submerged in a tank filled with fluid slime, and suction is then applied to the interior of the leaves. Applying suction to a filter completely submerged is to form an enveloping case or cake of a pasty nature in the filter. As the cake builds up it develops a thickness and compactness which gives the entire cake a capacity of uniform resistance to percolation. For so long as the resistance is not uniform the consequent increased rate of deposit at that point would set up and continue until the rate of flow there became equal to the rate of flow at all other points. The significance of this uniform resistance capacity of the cake, and that it was obtained by filter submergence, is stated in the specification where the patentee, in order to show that after the filtering process is completed an entire enveloping cake of uniform resistance capacity can be simultaneously discharged by compressed air, says that such action is owing to the process having produced an enveloping cake of uniform and even resistance capacity. His language is:

"In order to effectively discharge the encrusted slime from the filter by the agency of compressed air, it is important that the slime be in the form of a compact layer of requisite resistance and of sufficient thickness, because otherwise when the air-pressure is applied portions only of the slime are blown off, thereby relieving or reducing the air-pressure and rendering it ineffective for the removal of the slime which remains and necessitating the use of other means—such as scrapers, brushes, and washing—for the complete cleaning of the filter surface. This difficulty is wholly overcome in my process by immersing the filter into the tank containing the slime in suspension and depositing it in the manner described, the effect of which is to automatically deposit the slime in a homogeneous layer, as will be readily understood. Hence, when the slime has been thus deposited to the requisite thickness the compressed air does not blow holes in the layer of slime and only partly clean the filter, but it strips off the entire layer of slime and effectively cleans the filter without the use of auxiliary cleansing mechanism."

Indeed of the fact that the result of subjecting a submerged filter to suction is an enveloping cake of uniform and even resistance, there can be no doubt under the proofs in the case. To question it is to dispute the operation of the laws of nature. In his 'Cyaniding Gold and Silver Ores' (edition of 1907), H. F. Julian, who was called as an expert by respondent, and nowhere questions his prior statement, in describing the advantages of submerged-leaf filters subjected to interior suction, says:

"One of the chief characteristics of this class of filters is that during the formation of the cake, if the resistance to percolation should vary at any point over the filter surface, an adjustment immediately sets in, owing to the parts of the greatest permeability taking on the deposit quickest. This increases the resistance at those points until it brings the rate of percolation equal all over the cake. Washing out the dissolved metals is then done uniformly."

And in his testimony C. F. Chandler, the distinguished scientist and expert for respondent, says:

"Uniformity of resistance of the cake is the natural result of the laws of filtration. Increasing thickness of cake at any one point greatly reduces rapidity of filtration at that point and thus equalizes the thickness of the cake. When the thickness of the cake reaches a point at which filtration becomes very slow, the continuance of filtration at one part a little longer than at another part will not make any material difference in thickness; even doubling the time of filtration would have little effect if filtration is carried to the point of nearly maximum thickness and resistance, a condition which the Moore patent seems to indicate is desirable."

In view of these well understood natural laws of filtration and of the subject matter of the specification and the application of the

process as illustrated by description and drawings, there can be no question that, to those skilled in the art, the cake of uniform and even resistance produced by Moore's process is aptly described by him in his specification as "immersing the filter into the tank containing the slime in suspension and depositing it in the manner described, the effect of which is to automatically deposit the slime in a homogenous layer, as will be readily understood."

The specification in the language following discloses an optional additional step in the process, which step is made an element in the claims herein in controversy. The proofs show that in this step important results are obtained in that substantial quantities of the gold-carrying cyanide solution still remaining in the cake are recovered and that this recovery is due to the uniform and even resistance capacity conferred on the cake by the disclosed process.

"However, this cleaning step of the process need not be taken until an intermediate auxiliary step has been performed, which consists in introducing the element 2, after having been coated with the solids, into a tank 7 of water, the drawing or sucking operation of pump 5 being continued while the element 2 is being subjected to the said water-bath. When this step is employed, the next succeeding is the operation just described. It will be obvious that the water bath may be employed or not, as described, the same being preferable when the filter is used for filtering precious ores, the said step tending to wash out the remaining metal held in solution or solvent thereof within the solids coating the filtering device."

It is contended, however, that the Moore patent is invalid by reason of the disclosures of the prior art. But in our view this contention is based on a failure to recognize the true significance of what Moore really did. Practically his problem was to make commercially possible the recovery of a minute amount of valuable metal from a large quantity of mud. Of the fact of the metal being there, there was no doubt, for that fact, and indeed that it was possible to extract it, the tedious and costly method of laboratory filtration showed. It suffices to say that no one of the numerous patents cited did such work, used such process, or effected such results, and if none of them led their inventors or users to the use of any process whereby such work could be done, or even led to a suggestion in their descriptive matter of the possibility of the use of any such process as Moore's, it follows they taught Moore no more than others. So far as the patent here in question is involved, Moore's disclosure was the process he originated and not the machine with which he illustrated the use of his process in accordance with the statutory requirements that he file a written description "of the manner and process * * * of using it." To find, therefore, here and there in prior patents, and disassociated from each other, all the mechanical appliances of the combination ap-

paratus which Moore thus illustratively used is not to prove that Moore's process is not original.

Viewed from a patent standpoint, the significance of a machine lies not in its form but in the principle on which it works, as will be seen in the requirements of 4888 "in the case of a machine, he shall explain the principle thereof." It suffices therefore to say that very few of these patents are even for a process, and as none of them operated on the principle or process of Moore, they cannot be held to forestall or minimize the originality of Moore's subsequent disclosure. And in giving these patents their due relation to Moore's disclosure the fact must not be overlooked that the slime problem which Moore solved only came into existence from the use of the cyanide process, which began, as we have seen, about 1887. It will therefore be manifest that no patents preceding that date and none subsequent thereto which did not apply to the cyanide process, were calculated to solve the cyanide percolating difficulties, that arose in the use of that process. So also, to say that following prior laboratory practice, it was possible to leach and extract the unrecovered ore left in a pasty mass by the cyanide process is not to destroy Moore's patent; for this is to lose sight of the practical wroking value of Moore's process as a workable economic treatment, as compared with theoretical possibility of laboratory practice. By repeated dilution the laboratory could and, we will assume, did recover with practical completeness all such unrecovered metal, but this has been done with an expenditure of time, labor and expense out of all proportion to the value of the metal. When therefore Moore disclosed a process by which such recovery was made enormously profitable and by which he turned a dump heap which under all known processes, machines, and laboratory methods was worthless into profitable ore, we are constrained to give little weight to the suggestion that his process was either anticipated, a mere advance incident to the art, or involved no invention.

So, also, it is said that the step described in his claim, viz.: "further impoverishing the solids by a cleansing operation," was merely the washing or dilution of the prior art. Considered in its literalism and in isolation, such contention may seem plausible, but considered as a step in Moore's process it takes on a new significance and value. Bearing in mind that in the prior steps of the process covered by this claim, the completely submerged filtering medium has formed a cake of uniform resistance to all points and by reason of such uniformity the solvent has percolated and is permeating the cake, it follows that the step which follows is a non-diluting and bodily displacement of the solvent and not a diluting intermingling. This displacement in contrast with dilution, not only saves the time and expense of repeated dilution and refilling, but also obviates excessive dilution of the solvent solution and the necessity of rehandling large volumes of diluted solutions in the recovery of the metal. It seems, therefore, that the "further impoverishing the solids by a cleansing operation" of Moore's process, owing to the prior step whereby a cake of even and uniform

resistence is secured by submergence, is not a mere washing or diluting step, but is one wherein there is exerted a uniform pressure or pushing action through the entire cake surface, thereby in effect advancing a wall of water pressure to force ahead of it from the cake the value-bearing solvent liquid and leave in the cake an equal volume of non-value-bearing water. This final result is secured by first having built up a cake of uniform resistance to solvent fluid flow, and, secondly, by again submerging the filtering medium and its built-up cake in the non-value-bearing displacing fluid. By this displacement by pressure difference only Moore pushes ahead instead of washes through the cake a contained-metal-carrying fluid. As showing the practically complete metal extraction by the Moore process, we restrict ourselves to the uncontradicted testimony of results at a South Dakota mine, where the original slime contained gold at the rate of $7.90 per ton of dry slime. After filtration alone the cake still contained $2.75 per ton of dry slime. After being then subjected to the displacement step there was left in the cake but 4c. of gold per ton of dry slime.

Being of opinion, therefore, that Moore's process was novel, useful, and inventive in character, his patent is valid, and we next turn to the question of infringement.

As claims 4 and 5 furnish sufficient basis for deciding that question, so far as the respondent's device is concerned, and as some questions, not necessary here to be decided, exist as to claim 10, we restrict ourselves to a consideration of claims 4 and 5. In considering the question of the infringement of a process patent, it must be borne in mind that the monopoly secured by the claims is, generally speaking, a monopoly of the process, and the test of infringement is whether such process is utilized by the infringer. As the apparatus shown in a process patent is only to show that the process may be practically applied, it follows that such illustrative apparatus does not limit the process patent to that type of machine alone. If that were the case a process patent would be of little value. So distinctive and separate in the patent law are process and apparatus for utilizing such process that where, after a patent for a process by one inventor a second inventor might patent a novel apparatus for utilizing the process, the situation would arise that the inventor of the process could not employ his process in such machine without license from the machine patentee and the latter could not use the process in his machine without license from the process patentee. It will therefore be evident that the test of process infringement is not the similarity of apparatus, but rather whether the apparatus, no matter what its form, utilizes the process. Tested by this standard, it is clear to us the respondent's device infringes. In form the particular apparatus shown in Moore's patent and the apparatus of respondent vary in the number of tanks, in the differences between changing the fluid which envelops the filtering medium, as in respondent's device by allowing the filtering medium to remain stationary in one tank

while the submerging fluid is first drawn off and the second submerging bath is then drawn into the same tank, while in Moore's the filtering medium is raised from the submerging bath in the first tank and then lowered into the second submerging bath in a second tank. But this difference in numbers of tanks and of respective withdrawal and replacing of different baths in no way affects the identity of the process, for it is manifest that Moore could in his patent specification have shown the use of his process, just as well by using respondent's apparatus, had he known of it, as his own. Both alike use the principle of submergence and intraleaf suction to create the uniform and even resistance of the cake and both alike use the principle of intra-leaf pressure, Moore using air and the respondent water, to shed the uniformly resisting cake from the filter. For the mere fact of the output being carried off as a dry product in Moore's case to a dump heap and in respondent's in fluid form to a slime pit, does not go to the substance of the process. In the essentials that involve the invention the two are alike. Had an apparatus such as respondent's, been in use prior to Moore's there would have been no invention either in the process or in the apparatus shown in Moore's patent. And what, if preceding a patent, would have anticipated it, equally infringes if subsequent. We therefore hold the fourth claim is infringed.

As we are of opinion and so find from the proofs that there is in respondent's device a "removing the medium while continuing the drawing action," the fifth claim is also infringed.

The decree of the court below is therefore reversed and the cause remitted with instruction to enter a decree adjudging claims 4 and 5 valid and infringed and for such action by that court in the way of injunction and accounting as it shall deem fitting.

RECORDING GAUGE FOR FILTER OPERATION

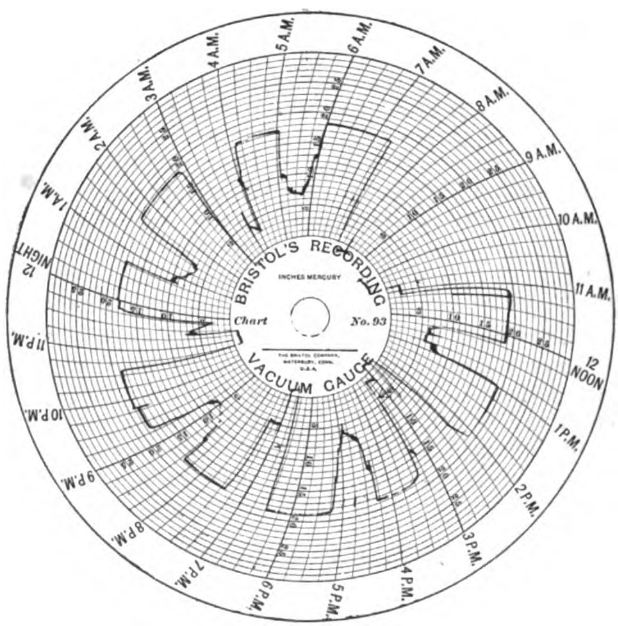
STAFF CORRESPONDENCE

(December 14, 1912)

The scheme of treatment at the Montana-Tonopah filter plant involves dumping 150 tons of ore per day into a steel ore-bin, from which it is drawn out, crushed by a No. 5 type K Gates, elevated to a revolving trommel, and the oversize reduced to 1¼ in. by two No. 3 type D Gates machines. A 14-in. belt conveys the broken ore to the mill-bin, from which it is fed to forty 1100-lb. stamps, crushing through 20 to 25-mesh screens in hot cyanide solution. There is about 3% of pyrite in the ore, and nearly one ton of concentrate is caught daily on 8 Wilfley tables. This is dried to 5% moisture and shipped to smelters. Pulp from the concentrators is lifted by two bucket elevators to two Dorr classifiers, which work in the closed-circuit system with two 5 by 22-ft. tube-mills. The classifier overflow goes to three settlers or dewaterers. Thickened

slime is then pumped to six Hendryx and one Trent agitators for 52 hours' agitation in a 5-lb. cyanide solution. Live steam is introduced here and the temperature kept to 110° Fahrenheit.

The vacuum filter plant consists of two filters with 100 leaves in operation, each 5 by 10 ft. The chart reproduced is a complete record and check on the work done in this department. The Montana-Tonopah mill is at an altitude of about 6500 ft., and the possible vacuum is 22 in., while from 18 to 20 in. is maintained during filtration. A study of the recording gauge shows, for instance: At 7 p. m. the filter tank was filled with slime, and formation of cake started, the vacuum keeping at 5 in. for about 25 minutes, then



rising to 18 in., remaining steady at this for 60 minutes, when the cake was formed, and excess slime pumped out by a centrifugal pump to the stock tank. From 9 p. m., when the tank was filled with weak solution for washing, the vacuum averaged 18.5 in., dropping momentarily to 16.5 in. at 9:50 p. m., until at 10:35 p. m., when treatment was finished, suction ceased, and the cake was blown off and discharged. The complete cycle, therefore, was 3 hours and 35 minutes. Silver and gold are precipitated by zinc-dust, supplied by an adjustable feeder, which also has an arrangement to prevent the dust from setting hard during feeding. An extraction of 93% is obtained at a total cost of $2.99 per ton.

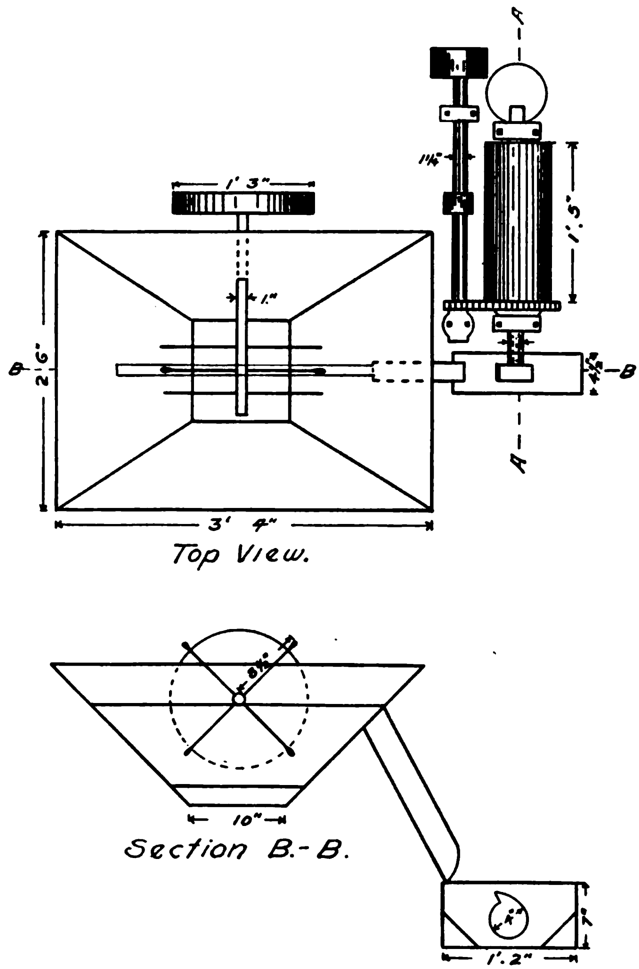
PRECIPITATION AND CLEAN-UP

AUTOMATIC ZINC DUST FEEDING

(February 10, 1912)

The Editor:

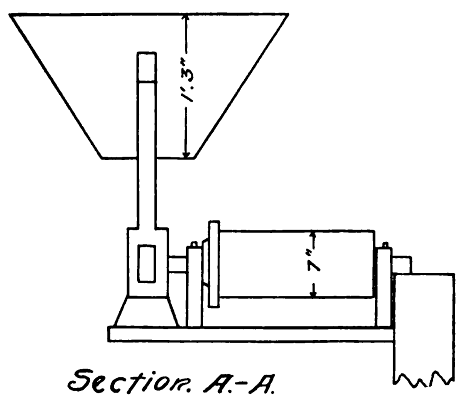
Sir—The drawing illustrates an automatic zinc dust feeder that is being used with success at the Palmarito mine, Mocorito, Sinaloa, Mexico. It seems to possess several points of superiority over any



feeder now in use. These points are: First, a positive continuous feed that is easily and quickly regulated at the will of the operator; second, extreme mechanical simplicity; third, practically automatic action. Zinc dust, owing to its hydroscopic properties, very readily absorbs moisture from the atmosphere, and the greater the percentage of moisture in the dust the greater its tendency to pack, and

therefore the more difficulty in securing a positive feed. This was the problem to overcome here. At the low altitude of the mine, only about a hundred feet above sea-level, the air contains a great deal of moisture, and no matter how tightly closed the zinc container, it would absorb a high percentage of moisture. The feeder overcomes this difficulty and gives a positive continuous feed under varying degrees of moisture.

It may be briefly described as follows: A rectangular box, with sides sloping at an angle of 60°, made of 1-in. material lined with galvanized iron, having a central feed-compartment or slot, about 1 in. wide, that reaches to within 1½ in. of the bottom of the box. This space is left so that the dust may enter the central feed-slot from each side, but not in such quantity as to offer any resistance to the revolution of the spokes that elevate the dust. An axle, 1½-in. diam., has attached to it four or more steel rods or spokes, ⅜-in. diam., with recessed openings at the ends shaped so as to



Section A-A.

form an elevator bucket to raise a definite quantity of zinc at each revolution. This shaft revolves at a speed of 15 or 20 r.p.m., not fast enough to cause the buckets to throw their contents by centrifugal force, but allows them to discharge by the force of gravity alone. The buckets are recessed in such a manner that they will not discharge their content until past the center of gravity. The rods or spokes are flattened to knife-blade thinness, so that no zinc will adhere to them in their passage through the dust. These buckets discharge into a chute, also made of galvanized iron set at such an angle that the zinc does not stick, but runs into a continuous stream into the spiral feed of a miniature tube-mill. We have adopted Yaeger's idea of a small tube-mill to make the zinc emulsion, and find that it works well. A small stream of water washes the zinc into the feed of the tube-mill from the point where it strikes on leaving the chute.

The spokes radiating from the axle are threaded, and screw into the axle, so that in order to regulate the feed it is only neces-

sary to take out a spoke and screw in another having a larger or smaller bucket, as may be needed to increase or diminish the feed.

To provide against the zinc in the container packing so that it will not run into the central feed-slot, a ⅜-in. steel rod slightly flattened at the ends is run through the axle on each side of the central feed-slot, and these rods serve as stirrers of the zinc and effectually overcome the difficulty of the zinc packing. A pulley on the shaft running the tube-mill connects with a pulley on the axle of the feeder and operates the feeder.

The container, as designed, easily holds 112 lb. of zinc dust, the amount usually packed in the sealed boxes in which it is marketed, and a cover can be provided for the feeder, thus preventing oxidation while in operation. Thinking that this device may be of use to the many cyanide operators throughout the country, I cheerfully offer it for their benefit.

A. W. MORRIS.

Mocorito, Sinaloa, December 31, 1911.

ZINC DUST TESTS

By W. J. SHARWOOD

(May 11, 1912)

*The increasing interest in zinc dust as a precipitant suggests the publication of details of a few tests which have proved of some use in examining and comparing samples. Unfortunately no definite criteria have as yet been established for valuing zinc dust for precipitating purposes, but the following will serve to give some idea as to the availability of a sample for the cyanide plant. It should be dry and fine; nearly all, say 95%, should pass a 200-mesh sieve (aperture 0.003 in.), while very little, say not over 1 or 2%, should remain on 100 mesh (0.006 in.), and practically none on 50 or 60 mesh. It should not show any signs of caking, or contain lumps which do not break up at once when shaken on a sieve. Generally speaking, an ash-gray color is a more favorable indication than a decidedly blue or whitish cast; but color cannot be taken as a positive criterion. The presence of a little lead, say 2 to 3%, is an advantage. Laboratory tests made with pure zinc, and with equal weights of alloys containing up to 5% lead, in the same state of fine division (filings between 0.006 and 0.003 in.), have invariably shown more rapid and more nearly complete precipitation of gold and of silver in the case of the lead alloys, although the weight of actual zinc present was less. The presence of a little zinc oxide is apparently without detrimental effect, except as diminishing the percentage of active metal. Attempts to remove the oxide by solvents inevitably lead to the dissolving of more or less metallic zinc, and possibly to the destruction of the finest particles. No injurious effect has been traced to the small amount of cadmium usually present, and other impurities are generally negligible in amount. A number of samples which have proved satisfactory in

*From the *Jour.* Chem., Met. & Min. Soc. of S. A.

actual use have shown from 85 to over 95% metallic zinc (that is, zinc in the metallic state), most of them over 90 per cent.

Various published analyses have shown metallic zinc ranging from less than 30 up to 92%, zinc oxide from traces to over 50%, and cadmium from traces up to 1.3% in Belgian, and 1 to 2.5% in Silesian samples. A number of analyses from various sources are to be found in Schnabel's 'Metallurgy' under Zinc, Ingalls' 'Metallurgy of Zinc and Cadmium,' and in 'Mineral Industry,' Vol. II, p. 258. Some of these show carbon up to 3 or 4%, and silica or insoluble up to 9 or 10%, but in general very little of either will be found, and only traces of iron or arsenic. A few of these analyses are summarized in the annexed table, together with incomplete determinations of some samples used in gold precipitation.

Typical Analyses of Zinc Dust[1]

		Metallic zinc.	Zinc oxide.	Lead.	Iron, cadmium, carbon and insoluble.
1.	Belgian	91.5	Little	0.5	...
2.	Belgian	88.74	6.60	2.5	2.1
3.	Belgian	79.16	11.16	1.9	7.2
4.	Average Belgian	84.5	9.3	1.6	4.0
5.	Silesian	84.5	4.9	4.3	6.2
6.	Silesian	88.5	7.4	2.0	...
7.	Silesian	....	88.2	4.0	3.0(SO_4 1)
8.	American	29.6	57.5	Trace	(Insol. 9.6)
9.	American, good quality	96.0	...	...	...
10.	European, imported into U. S., good quality	90.9	4.0	2.7	...
11.	European, similar, fair quality	85.0	...	2.0	...
12.	European, similar, fair quality	94.6	3.7	0.2	(Iron 0.3)
13.	Used at Deloro, Canada	...	3 to 5	1.74	...

Various authorities state the specific gravity of zinc as from 6.85 to 7.21, of zinc oxide from 5.6 to 5.78, and zinc carbonate 4.3 to 4.5. Samples of zinc dust have shown results from 6.82 to about 7.0, while the material itself weighs approximately 2 to 2.5 gm. per cubic centimetre, or about 120 to 155 lb. per cubic foot, depending largely upon the extent to which it has been shaken down. No connection has been proved between the apparent density and the quality, but at least one 'heavy' sample was found to give unsatisfactory precipitation.

The following serves to illustrate the desirability of avoiding contact with moisture or moist air in the transportation and storage of this material. On a carload of imported zinc dust used some years ago in Montana the railroad charges were found to be based on a considerably greater weight than was indicated by the weights of the several casks invoiced as originally shipped from Europe. The total weight found by the daily use of the material bore out

[1]No. 1 to 4 by Firket, 5 to 7 by Steger, 1 to 7 quoted by Ingalls; 7 and 8 from Schnabel, intermediate products in smelting; 10, analysis supplied by importers, quality excellent; 13, Harland, *Jour.* Soc. Ch. Ind., 1897, p. 968; 9, 11, and 12 original; No. 9 to 13 used in cyanide precipitation at various plants.

the railroad figures, and confirmed the suspicion that the discrepancy was due to oxidation from contact with moist air. This was further proved by analyses for oxide and metallic zinc, a sample taken near the staves of one cash showing about 10% more oxide than another from the middle of the mass. In another case, where very bad precipitation was noted, the material from the bottom of one barrel was lumpy and yielded 39% zinc oxide, while a sample from near the top of the next barrel opened contained 13.5 per cent.

Estimation of Lead in Zinc Dust.—Weigh 10 gm. into a large beaker (400 to 600 c.c.), moisten with water, then add 200 c.c. water and stir well, then 10 c.c.[2] strong sulphuric acid, stir well and cover. Stand beaker on hot plate and warm to 60 or 70°C.; if action is very slow at the start add a single drop of dilute platinic chloride solution; stir at intervals. After action becomes slow add 10 c.c more acid and continue warming until nearly all the zinc has dissolved. Decant the solution upon a filter, wash the residue in beaker two or three times with water, passing washings over the filter.[3] Rinse residue back from filter into beaker. Dissolve residue with a little nitric acid,[4] transfer to a casserole, and rinse beaker into casserole. Add 5 to 6 c.c. strong sulphuric acid and evaporate to dense fumes. Cool, dilute with water, stir well, cool thoroughly, filter, wash well with water containing 5% sulphuric acid.[2] Dissolve the lead sulphate in hot ammonium acetate solution, and wash filter thoroughly with the same solution. Dilute this solution with hot water (about 80°C.) and titrate the lead with standard ammonium molybdate, using 1% tannin or ferrocyanide solution as outside indicator.

The molybdate is standardized on pure lead foil, treated in the same way as the lead residue from the zinc dust. If the ferrocyanide titration is substituted for the molybdate method, the same standard solution will serve for both lead and zinc.

Estimation of Zinc Oxide in Zinc Dust.—Approximate method based on solubility of zinc oxide in ammonium chloride and ammonia. Solution required, 250 c.c. water; 70 gm. ammonium chloride; 150 c.c. strong ammonia water, sp. gr. 0.90.

Method.—Weigh out 1 gm. zinc dust into a stout test-tube holding 35 to 50 c.c. Use a disc of soft rubber packing $^1/_{16}$ in. thick as a cover for the tube, or use thumb as cover while shaking. Add 25 c.c. of the prepared solution, cover tube, and shake well for exactly five minutes. Agitation must be violent enough to insure

[2]The use of 25 c.c. of strong hydrochloric acid, in place of 10 c.c. sulphuric acid, gives equally satisfactory results.

[3]The two filtrates from the lead and lead sulphate contain the iron, cadmium, and zinc; they may be mixed, made up to a definite volume, and these metals determined in suitable aliquot portions.

[4]If any considerable residue (silica, carbon, etc.) remains at this stage after treatment with nitric acid, it may be filtered off, washed thoroughly, and determined. If in this case a white residue remains after washing, it should be treated with hot ammonium acetate solution, and this should be added to the acetate solution obtained later.

all lumps being broken up. Throw mixture promptly on a 9 cm. or 11 cm. filter, rinse tube, and wash filter with hot water, receiving in a 200 or 250-c.c. beaker. Dilute filtrate with hot water to about 100 c.c. Add a drop of phenolphthalein indicator, then HCl till about neutral, then 5 c.c. strong hydrochloric acid. Heat to 80°C. and titrate zinc with standard ferrocyanide, using uranium acetate or nitrate as indicator. Zinc found in solution $\times$ 1.245 = zinc oxide in 1 gram.

This method is rapid and gives fairly approximate results, which are closely comparable if constant conditions are maintained in all tests. The results are slightly high, for the reason that metallic zinc is itself slightly soluble in the solvent employed—but there is no known solvent for the oxide which will not also attack the metal. Tests with freshly prepared zinc filings, passed through a 200-mesh sieve, show that the error from this source is negligible if only five minutes' contact is allowed with the above solution. Carbonate of zinc dissolves in the same way as the oxide.

Estimation of Total Zinc.—Dissolve 1 gm. of dust in dilute hydrochloric acid, heating moderately. Make up to 200 or 250 c.c. in an accurate flash, remove 50 c.c. with an accurate pipette. Add 5 c.c. strong hydrochloric acid and about 2 gm. ammonium chloride, dilute to about 200 c.c. with hot water, heat to 80°C., and titrate with ferrocyanide. The ferrocyanide solution is standardized on pure metallic zinc, dissolved in hydrochloric acid, and treated as above.

Determination of Metallic Zinc in Zinc Dust.—Many methods have been worked out, most of which are not altogether satisfactory. Those which depend on measuring the volume of hydrogen evolved by the action of acid are hampered by the slow rate at which zinc dust dissolves in dilute acid, and by the readiness with which hydrogen diffuses through rubber tubing and leaky connections. A number of the methods proposed are given in Sutton's 'Volumetric Analysis' (10th ed., pp. 382-383).

I. Very good results are obtained by determining (1) *total zinc* and (2) *zinc in the form of oxide,* by the methods given above. The difference is assumed to be *metallic zinc.* This has the advantage of requiring only one standard solution—potassium ferrocyanide—for both titrations.

II. Reduction of chromic acid, by subjecting the zinc dust to the action of potassium bichromate acidified with sulphuric acid. The bichromate should be present in considerable excess of the calculated amount necessary for the zinc used. The residual chromic acid is determined by titrating an amount equal to that originally taken, and the solution after the zinc has acted, or aliquot parts of each, either:

(1) by means of KI, titrating the liberated iodine with thiosulphate (Sutton, p. 383); or

(2) by means of a standardized solution of ferrous sulphate;

(3) by adding an excess of a standardized solution of ferrous sulphate and completing titration with standard bichromate.

III. Reduction of ferric chloride or sulphate and titration of the ferrous salt obtained by standard permanganate or bichromate. Sutton (p. 383) recommends using 0.5 gm. zinc dust with 7 gm. of ferric sulphate and titrating one-fifth of the solution with permanganate. Presumably 14 or 15 gm. of ferric alum would answer equally well for this purpose.

IV. The reduction by the zinc dust of a solution of iodine in potassium iodide has given fair results, but it is only safe to use small quantities, say 0.1 or 0.2 gm., titrating the residual iodine with thiosulphate.

The following are results obtained with a sample of zinc dust:

	Per cent.
Method I:	
Total zinc	91.6
Zinc as oxide (and carbonate)	12.9
Metallic zinc by difference	78.7
Method II:	
Metallic zinc by reduction of bichromate (3)	79.6
Method III:	
Metallic zinc by reduction of ferric alum	77.1

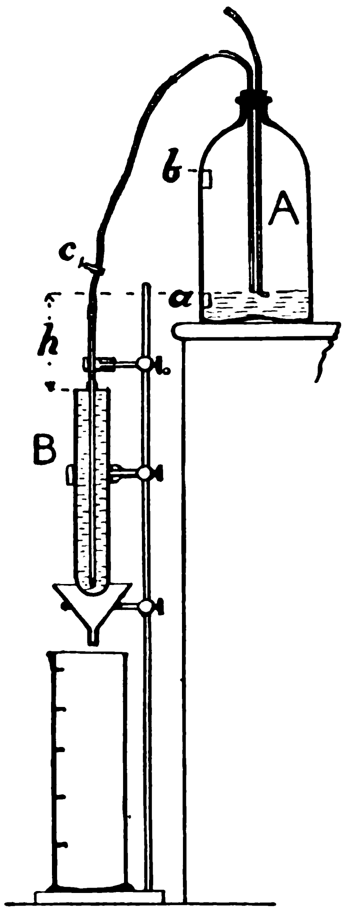
Whatever *reduction* method is employed for direct determination of the metal, the metallic zinc can be calculated thus:

1 c.c. $N/10$ standard solution = 0.00327 gm. zinc.

Determination of Precipitating Efficiency of Zinc Dust.—Solution required. A solution of potassium silver cyanide is prepared by dissolving 10 gm. silver cyanide (AgCN) and 5 gm. of 99% potassium cyanide in a little water and diluting up to 1000 c.c. It is then adjusted by addition of a little more KCN or AgCN until the solution indicates from 0.12 to 0.15% *free* KCN by titration with standard silver nitrate. The titration is best made by using a 10 or 20-c.c. sample, adding 1 c.c. 2% potassium iodide and a slight excess of ammonia; the end-point is then sharper. Or 15 gm. of pure crystallized $KAg(CN)_2$ may be dissolved in a litre of water and 1.5 gm. KCN added.

Method: Weigh out 0.5 gm. zinc dust into a 300-c.c. beaker. Add a few cubic centimetres water and stir till zinc is well mixed, then pour in 250 c.c. of the prepared solution, stirring vigorously. See that all lumps are broken up, and continue stirring for fully five minutes. Stir occasionally (at least every 10 minutes) until the end of two hours from the addition of the solution. Then filter upon an 11-cm. filter, wash precipitate thoroughly, sprinkle with test lead, wrap it carefully in the paper, place in a scorifier with about 20 gm. test lead, burn paper cautiously in muffle, scorify 5 minutes, cupel at low temperature, and weigh silver. Milligrams silver obtained from 0.5 gm. zinc × 0.0606 = percentage precipitating efficiency.

By taking 0.303 gm. zinc dust instead of 0.5, each milligram of silver is equivalent to 0.1% efficiency. Instead of cupelling, the silver may be dissolved in dilute nitric acid and titrated with standard thiocyanate with ferric indicator, using the whole or one-half of the solution for titration. C.c. $N/10$ KCNS used for 0.5 gm. + 0.654 = percentage efficiency.



CLASSIFICATION APPARATUS

This test was devised by A. J. Clark in 1904, with the idea of utilizing a reaction analogous and taking place under conditions similar to those actually prevailing in the everyday use of the zinc dust in the cyanide plant. While the results obtained by this method are not entirely satisfactory, they have afforded, during seven years' experience, a better indication of the results to be expected in practical work on the large scale than those obtained by any other system of examination so far tried. Satisfactory samples of dust usually show efficiencies between 40 and 60%, and sometimes as low as 35%, 100% being the standard calculated for the complete replacement of pure zinc by silver. A better method—one which completely eliminates the personal factor in manipulation—is desirable, but a simple determination of the metallic zinc is not sufficient; the test must automatically take into account the physical condition as well. Attempts to carry out the test in a mechanical shaker, and determining the silver left in solution after filtering after a specified time of contact, have always given much lower results than are obtained as above.

Mechanical Tests of Zinc Dust.—Sizing Test: When sized with sieves of 100 and 200 meshes to the linear inch, using a sample of 20 to 50 gm., good samples of zinc dust usually show at least 95 and often 98% or more passing the 200-mesh, and not more than 1% remaining on the 100-mesh. This sizing is irrespective of caked lumps, which should never be found in freshly opened barrels.

Classification Test: This is based on the separation of the finest particles by a rising stream of water in a vertical tube, the velocity being about 1 cm. per second. It promises to throw some light on the fineness of division of the portion passing 200-mesh,

which, as above indicated, ought to include nearly all of the material.

A convenient arrangement is that shown in the figure. A 2500-c.c. acid bottle (A) is fitted with a 2-hole rubber cork, with a Mariotte tube reaching nearly to the bottom and turned at a right angle at the lower end, and a siphon outlet tube reaching to about the same level, thus giving a constant level water-supply. A mark is placed at *a*, the level to which the siphon empties the bottle, and another at *b* to indicate the surface when 2000 c.c. additional water has been put in. The working cylinder B may be a tall 250-c.c. measuring-glass, but it is better to use a large tube with rounded end, say 1.25 in. diameter by 12 or 15 in. long, with a ¼-in. glass tube placed centrally in it and connected with the siphon by a rubber tube with a spring clip c, both being fastened to a stand by clamps. A large graduated cylinder and funnel are placed below. The net area of the cross-section of the working cylinder is best obtained by sticking two labels or making two marks on it 20 cm. apart, then filling with water to the lower mark and measuring the number of cubic centimetres required to fill it to the upper, the central tube being clamped in place. Then the area of cross-section (= volume in c.c. ÷ height in cm.) is obtained once for all. To adjust the velocity fill the bottle to the upper mark, fill the working cylinder, open spring clip and take time with a stopwatch while 1000 c.c. flows out.

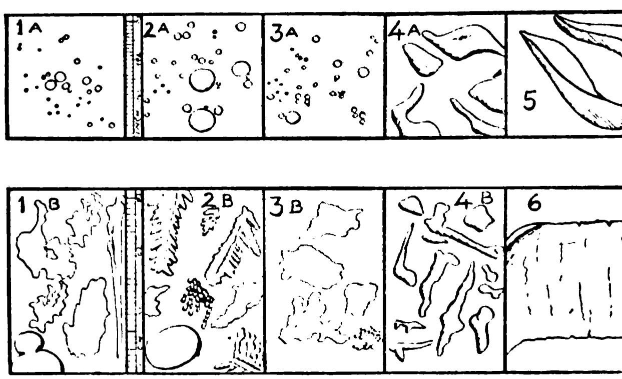
$$\text{Then } \frac{1000 \text{ c.c.}}{\text{Area} \times \text{No. sec.}} = \text{velocity in cm. per second.}$$

By raising or lowering the working cylinder the velocity can be adjusted, the effective head *h* being the vertical distance of the top of the cylinder below the level of the Mariotte tube inlet, the velocity varying approximately as the square root of *h*. The clip on the siphon should be removed or kept wide open during the test, and care should be taken that the rubber connections are not sharply bent or allowed to obstruct the flow. When the point is found at which an upward velocity of exactly 1 cm. per second is obtained, or some point sufficiently near, it is well to mark the position of the cylinder and clamp or measure the height *h*, so that this position can be reproduced, all the other adjustments remaining the same.

Method: Put 20 gm. of zinc dust in the cylinder, moisten with a little water, then add 50 or 100 c.c. and stir till all air bubbles are removed, then fill cylinder with water. A wire stirrer is used, reaching to bottom. The bottle being filled to *b*, open clip and start the flow. Use the stop-watch as a check on the velocity, timing the first 1000 c.c. The wire stirrer must be used at first to prevent packing; most of the fine passes off with 500 to 1000 c.c. of water, but it is best to let 2 litres flow through. Then allow the remaining material to settle a few seconds, rinse into an evaporating-dish or casserole, decant off water, wash twice with a little alcohol,

dry, and weigh the residue. This may be tested with 100 and 200-mesh sieve, if not sifted before making the test, and further examined with the microscope. By commencing the test with a velocity of say 0.8 cm. per second, and repeating with the same material at 1 and 1.2 cm., several distinct fractions may be obtained from the material passing 200-mesh. The comparatively coarse particles are of several kinds: (1) impurities, usually chips or sand; (2) round particles of zinc, or aggregate made up of round particles, looking like grape-shot: (3) feathery or fern-like aggregates of minute crystals of zinc, often forming very thin platy masses.

A separation of the coarser particles can, of course, be more simply made by repeatedly stirring the water in a beaker and decanting, but it is impossible to obtain comparable results unless a constant upward velocity is maintained in some such way as indicated.



MICRO-PHOTO OF ZINC DUST.

Description of Figures.—(1) Imported (German) zinc dust; satisfactory, precipitation good. (2) Imported (German) zinc dust; unsatisfactory, precipitation not good. (3) Domestic (U. S.) zinc dust; satisfactory, precipitation good. (4) Granular zinc. (5) Zinc filings, passed 200-mesh sieve. (6) Zinc shaving, extra fine about 1/50 in. wide.

The accompanying figures illustrate some of the variations shown in the appearance of certain samples of zinc dust when examined under the microscope. The upper row, highly magnified, show the appearance of the medium portion of the 'through 200' fractions of three such samples, the very finest part (under $\frac{1}{20,000}$ in.) having been removed by classification. As may be seen, these are almost perfect spherules, and under the highest powers appear to have clean smooth surfaces, and to consist of nearly pure metal. The coarser particles, over 0.003 in. remaining on a 200-mesh sieve, are shown below, and vary greatly in appearance. In several of the more satisfactory samples, illustrated by No. 1 and 3, many of these larger particles have a rough coke-like surface, but some generally exhibit branching crystalline, fern-like aggre-

gates, and other aggregates resembling miniature bunches of grapes. It is possibly only a coincidence that the unsatisfactory sample (No. 2) had an unusually large proportion of the fern-like forms and also showed more relatively large spherules in the 'through 200' portion. The extreme minuteness of the fine is shown by comparison with the equally magnified filing (No. 5) which had passed a 200-mesh sieve, and the granulated zinc, also passing 200-mesh (4A), the method of preparation of which is not known, but which had a very low precipitating efficiency.

Surface Exposed by Zinc Dust.—As the efficiency of this precipitant is mainly due to its fine division or extensive surface, some calculations of the latter may be of interest. To calculate the surface exposed by zinc dust, use may be made of the fact that the total surface exposed by one ton (2000 lb.) of any material in uniform grains—either cubes or spheres—is $\frac{2304}{g\,d}$ sq. ft., g being the specific gravity of the material and d being either the edge of the cube or the diameter of the sphere, expressed in inches. Hence, the density of zinc being very nearly 7, assuming it to be in the form or spheres of a uniform diameter of d inch, then square feet surface exposed by 1 lb. $=\frac{0.165}{d}$.

Thus the portion which will *just* pass a 200-mesh ($d=0.003$ in.) sieve will expose only 55 sq. ft. per lb., while the material averaging only 0.0001 in. diameter—which constitutes a considerable proportion of the whole, as may be seen by a microscopic examination of the finer portion—exposes 1650 sq. ft. per lb. It is interesting to compare this with zinc shaving assuming this material to consist of a continuous rectangular strip, of width 'a' and thickness 'b', then:

Area exposed by 1 lb. zinc shaving $=0.055\ (\frac{1}{a}+\frac{1}{b})$ sq. ft.

If, instead of being rectangular, the cross-section of the shaving is a parallelogram of angle 'A', the above expression must be multiplied by cosec A; in this case the thickness b is measured at right angles to the width a.

For instance, with shavings $^1/_{32}$ in. wide and of various thicknesses, the surface exposed would be:

Thickness	1/800	1/1000	1/1500 in.
Sq. ft. per pound	45.8	56.8	84.3

The number of particles (spheres) of zinc dust in a pound, assuming the specific gravity as 7, is $\frac{7.544}{d^3}$. Hence, if the diameter is 0.001 in., 1 lb. will contain 7544 million particles, while at an average diameter of 0.0005 in. there will be 8 times as many. Thus

some idea of the intimate nature of the mixture can be obtained if an emulsion of zinc dust is prepared by distributing it with absolute uniformity through the solution to be precipitated. Suppose $^1/_5$ lb. of zinc dust is used per fluid ton (55,296 cu. in.) of solution, or 1 in 10,000 parts by weight, the diameter of the particles being taken as 0.001 inch. From the above figures it appears that each cubic inch will contain about 27,300 particles, if uniformly distributed, whence the average distance between the particles will be approximately $^1/_{30}$ in. If the particles are half this diameter and equally distributed, there will be 8 times as many to the cubic inch, and half the distance apart.

All samples shown in the upper row highly magnified (see scale); those marked (A) are carried off by a stream of water rising 1 cm. per second, but finest dust has been removed. Those marked B remain on a 200-mesh sieve and are less magnified, as shown by the scale beside 1.

In the October *Journal* of the Chemical, Metallurgical & Mining Society of South Africa, is a communication from M. Thornton Murray on this subject. The sample of zinc dust examined by him appears to have been somewhat coarser than the average, as it is usual to find in the material of less than 0.003 in. diameter a good deal of dust not over 0.001 in. diameter, and some considerably finer, so that I should be disposed to consider 0.0005 in. more nearly an average for this fraction.

Mr. Murray's assumed gravity (6.92) would give $\frac{0.1665}{d}$ instead of $\frac{0.165}{d}$ which I have assumed for the sq. ft. per lb., or approximately 1% more surface than my formula would indicate. Thus, instead of 144.8 sq. ft. per lb. as calculated by him for a mean diameter of 0.00115 in., my formula gives 143.5 sq. ft. Actually I have found a range from 6.82 to 7.0 in the specific gravity of samples of zinc dust tested.

Mr. Murray has, however, seemingly obtained his 'mean diameter' by multiplying the assumed diameter of each of his fractions by its percentage weight, and dividing the sum of these products by 100, which is not mathematically correct for the reason that there are far more particles in 1% of the finer grades than in 1% of the coarse,[5] considering a given weight of the total material. This makes his calculated mean diameter too high, and the resulting surface correspondingly too low. This may be seen by computing the surface for each fraction separately, as below, from Mr. Murray's data for percentages and diameters of the various fractions, the

[5]This is obvious from a consideration of an extreme case. Suppose a mixture of equal weights of bullets of two sizes, half the weight being balls of 1.0 in. diam., and half of 0.10 in. The mean diameter is not 0.55 in. (as would be found by calculation based simply on distribution by weight), since for every 1-in. ball there are 1000 balls of a diameter of 0.10 inch.

finest fraction alone giving a larger surface than he has computed for the whole.

Fraction between	Mean diam. of fraction (d inch)...	Sq. ft. for 1 lb. of fraction 0.165/d..	Per cent of fraction	Sq. ft. for 1 lb. of total zinc......	Sq. ft. for 0.90 lb. of metallic zinc....
0.0197 and 0.0100......	0.149	11.07	0.16	0.018	0.0162
0.0100 and 0.0060......	0.008	20.63	2.42	0.50	0.45
0.0060 and 0.0030......	0.0045	36.67	2.88	1.06	0.954
0.0030 and 0.0.........	0.00085	194.1	94.54	183.50	165.15
Total..............	0.0009	183.67	100.00	185.08	166.57

If the average diameter of the particles in the finest fraction be assumed at 0.0005 in. instead of 0.00085, the surface becomes approximately 330 sq. ft. for each pound of that grade, or 312 ft. for a pound of the total dust, or about 288 sq. ft. of zinc reckoning 90% of metallic zinc, the surfaces exposed by the coarser fractions being negligible in comparison. As others have pointed out in a previous discussion on fine grinding,[6] the results of the calculation depend almost entirely upon the average diameter assumed for the material passing the finest sieve used, which there is no means of precisely determining.

ZINC DUST PRECIPITATION

(August 3, 1912)

The Editor:

Sir—Errors of mill construction materialize when the operator is unable to obtain the results expected by the designer. Errors of detail become more serious as the run continues, and their cause generally proves to be the neglect of the constructing engineer, who has followed the so called 'general mill plans' too closely, instead of consulting the installation prints of apparatus supplied by inventors and patentees.

One of my most recent experiences was in Arizona. The Merrill zinc dust precipitation process was installed at the time of constructing the mill. Some curious non-results were obtained during the first attempts at precipitation. An inexperienced operator performed the operations as outlined and as universally adopted. Samples of the ingoing pregnant and barren effluent solution assayed the same, indicating absolutely no precipitation of the precious metals. A close regeneration was not expected from the weak NaCN solutions with little protective alkali containing an extremely small gold content, due to the adoption of the decant-

[6]See *Jour.* Chem. Met. & Min. Soc. of S. A., Vol. VII., pp. 120, 207, 265, 289.

ation methods before the completion of the vacuum-filter. But certainly some press extraction was anticipated even under these conditions. Laboratory tests resulted in the precipitation of the metals from the solutions with a longer contact than the installation would allow before reaching the filtering medium. The Bosqui dust-feeder and tank apparatus seemed a necessity. After these tests I inspected the plant, finding that a pressure of over 25 lb. showed on the gauge with the press filtering at but 50% of its rated efficiency. The filtering medium was light twill, but it was evident that the press was getting an oversupply of air, as the solution coming from the spigots was foaming and bubbling. The air-connection to the press was evidently closed, and at first I thought the pump was taking air at the plungers, until noting the arrangement of piping and position of valves. In connecting up the 5 by 7 triplex electric pump supplying the press, a by-pass was neglected, and the valve on the pump suction was placed on the tank side of the zinc dust feed-pipe. This valve was used in regulating the press flow and pressure, and naturally the pump took an enormous amount of air from the dust feed-pipe.

To overcome these errors and to lengthen the contact of the zinc with the solution, a return by-pass pipe was extended from a point near the press to the suction of the pump. A valve was placed on this line near the press to regulate the flow and pressure to the press, and a check-valve was put on the suction between the return pipe and dust feed-pipe to check the return of solution to the tank. Thus the circulation of dust and solution before filtration was secured. Results were obtained immediately after these changes, and the press was brought to a high efficiency. It seems obvious to me that by installing a pump of greater capacity than the press, with the use of this return pipe, increasing the velocity and friction within the supply pipe, the danger of precipitate accumulation within the pipes would be lessened and the agitation and longer contact beneficial, in some cases doing away with the Bosqui tank and feeder installation. In this case zinc shaving boxes were ordered to replace the dust apparatus by the local mine manager, of South African experience. The order did not pass the main office immediately, so the unnecessary expense was avoided. Think of the enormous bulk of zinc shaving necessary for the precipitation of 250 tons per day of 50c. solution, and the $4000 or $5000 gold absorption, compared with the simplicity of the zinc dust appliances.

E. S. PETTIS.

Mill Valley, California, July 11.

ANALYSIS OF ZINC DUST

(September 16, 1911)

Among the many methods of analyzing zinc dust, the Drewson method, recommended by Paul Speier, is one of the most convenient and most reliable. It is based upon the fact that finely divided

zinc reduces a solution of chromic acid (bichromate of potassium and sulphuric acid) without generating hydrogen. The process is carried out as follows: Dilute 40 gm. of molten bichromate of potassium to make 1 litre, and adjust the litre with a strongly acidulated ferrous sulphate solution by means of the ferricyanide of potassium volumetric test. Put 1 gm. of zinc dust into a glass beaker and pour over it 100 c.c. of the bichromate solution. Add, while gently shaking the beaker, 25 c.c. of diluted sulphuric acid in portions of 5 c.c. each, at intervals of from 8 to 10 minutes. Let solution stand for an hour, and then add another 25 c.c. of diluted sulphuric acid. Then put the whole mixture into a volumetric apparatus and dilute to make about 500 c.c. Then gradually add, while strongly stirring, ferrous sulphate solution (200 gm. $FeSO_4+7H_2O$ +25 c.c. H_2O_4 concentrated to 1000 c.c.) until one drop with ferricyanide of potassium solution produces a weak but distinct blue coloring. In order to determine the quality of the metallic zinc contents, the cubic centimetres corresponding to the iron solution which has been used have to be deducted from the bichromate of potassium solution, and the bichromate of potassium contained in the remainder has to be multiplied by 0.66113.

ZINC DUST SOURCES OF SUPPLY

Zinc dust used in cyanidation in this country comes mainly from Germany, though some is now being made at Pueblo, Colorado. The material is the 'blue powder' of the zinc smelters, and consists of finely divided metallic zinc in part covered by a film of zinc oxide. The individual particles are often in a state of tension as in 'Prince Rupert drops,' and hence the material is explosive under some conditions. Blue powder is a troublesome byproduct in zinc smelting, and since zinc dust is bought in carload lots by the companies that have adopted the Merrill process, there would seem to be an opening here for a good American business. The ideal zinc dust would consist of 97% metallic zinc and 3 of metallic lead. The freer it is from oxide the better, and it is particularly important that it should be even in composition. Dust from new sources should not be used without thorough preliminary tests.

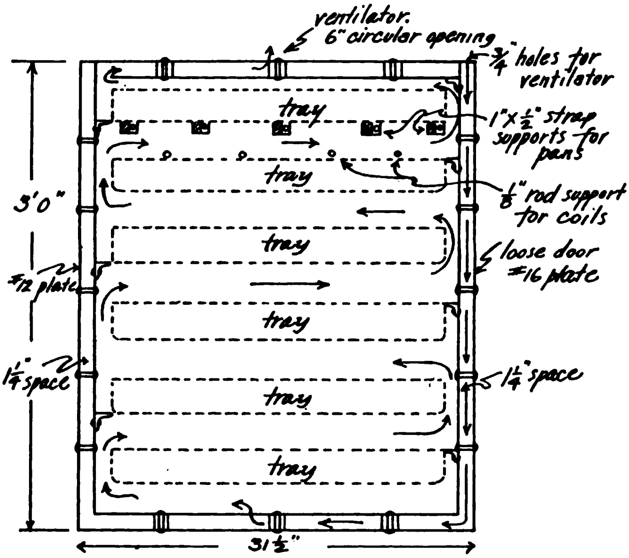
ELECTRICAL DRIER FOR ZINC PRECIPITATE

By DONALD F. IRVIN

(March 16, 1912)

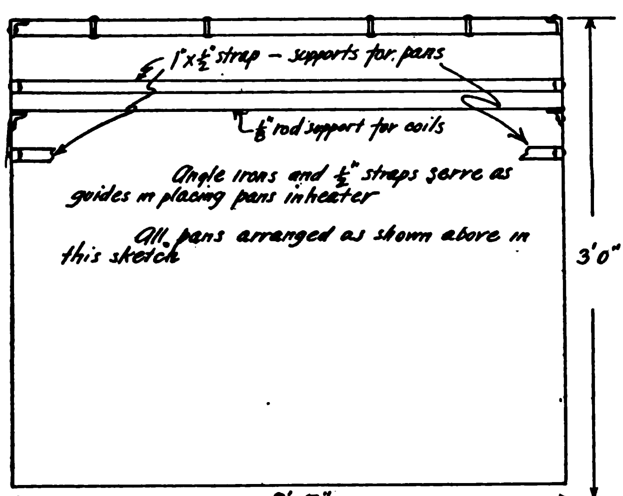
The preparation of zinc precipitate for smelting involves a preliminary drying, although if roasting is not desired, it is customary to charge the precipitate into the crucible with a higher mosture content than formerly practiced. Drying is done in various ways by direct heating, as in a coal-fired oven; or in a steam-bottom pan; or electrically. Some features of the last method are

that drying may be carried on without the dirt caused by stoking a furnace, and also with the minimum attention in the way of regulating the heat. Furthermore, heating can be carried on safely



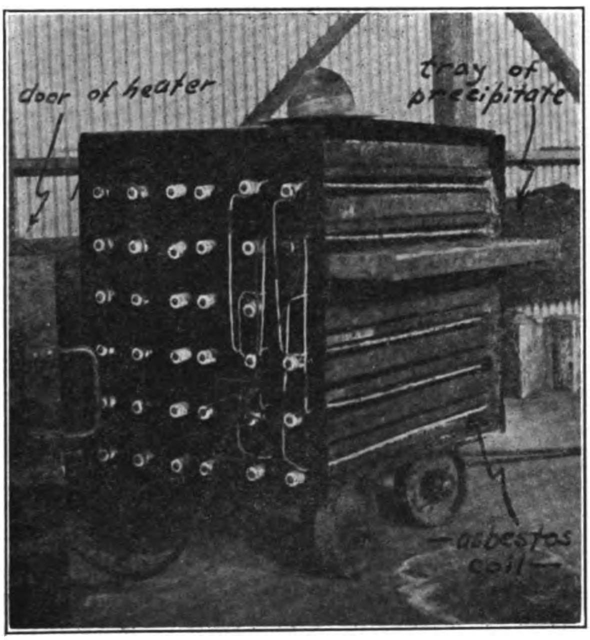
TRANSVERSE SECTION

at night, when attendants are usually off duty, a consideration of some value when melting is done on day-shift only. Thus, a small drying device does the work of one of twice the capacity, requiring constant watching.



LONGITUDINAL SECTION

A most satisfactory electric dryer is that installed for the El Tigre Mining Co. by D. L. H. Forbes. It consists of a series of sheet-iron trays, enclosed in a ventilated sheet-iron cage, the whole mounted on wheels for ease of movement. The accompanying illustration and sketches show the construction and wiring of the heater, which is simple and effective. This drier is used for the precipitate gotten from Merrill zinc dust precipitation presses, and no especial care is required in manipulation, save that time is gained by partly air-drying the precipitate before removing from the presses. Condensation of the rising steam occurs on the inner side

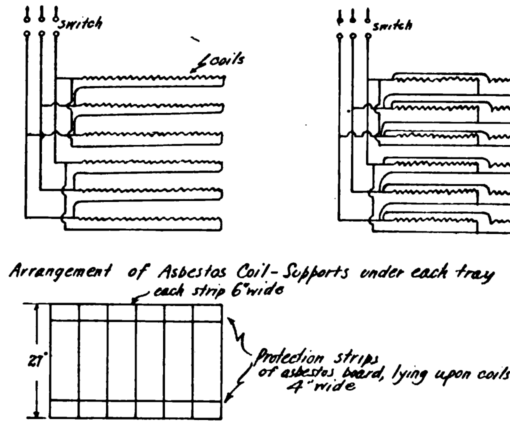


ELECTRIC PRECIPITATE DRIER

of top, making contents of top tray damp through dripping. The remedy for this is more ventilation, which must be balanced against undue loss of heat thereby.

The heating current is supplied from the regular 110-volt electric lighting circuit, and the resistance is furnished by coils of No. 18 hard-drawn iron wire. There are 1620 ft. of wire per phase, and in all, 4860 ft. is used. Each phase serves two trays, and each tray has 8 coils. Thus, each coil is wound with a little more than 100 ft. of resistance wire. This requires 30 to 35 amperes current at 110 volts. The capacity of this drier is about 175 kg. of precipitate, dried from the usual damp condition of fresh air-dried precipitate to absolute dryness in 24 hours.

Short-circuiting, resulting from the moist, dripping condition of the interior of the cage, is one feature to be guarded against. This may be prevented by smearing the coils with plaster of paris, thus protecting them from moisture. These wire coils are wound upon strips of asbestos board, 6 in. wide by 27 in. long, and ¼ in. thick, being supported by small longitudinal rods of ⅛-in. iron, as shown in the sketches. When first built, this dryer had glass tubing wound with asbestos wicking instead, but was soon changed to the present plan.



WIRING DIAGRAM

This is a very handy machine and saves considerable in first cost over the construction and maintenance of a brick oven drier. A steam bottom pan is not feasible at El Tigre, as all motive power is now electric, and the former boiler room is several hundred yards from the melting house. Also, steam losses in operation would be high. Comparisons of operating cost with other methods of drying cannot include the highly desirable features of the electric drier making for convenience, cleanliness, and safety; all of which incline millmen in its favor, especially after having used one.

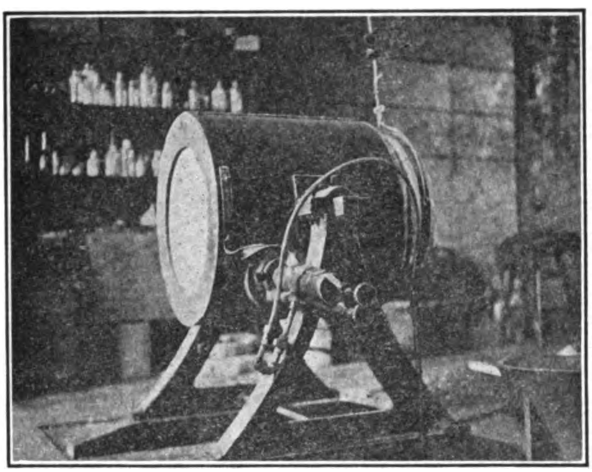
COMBINATION MELTING AND CUPELLING FURNACE

By Will H. Coghill

(June 17, 1911)

Coke-fired melting-furnaces of sufficient size to hold large crucibles have been used for several centuries. In operating this type of furnace the melter is obliged to expose himself to intense heat while drawing the crucible from the furnace and pouring its contents. As the result of the recent development of the cyanide process in which the 'clean-up' must be given a preliminary treatment and melted, the furnace has been greatly improved, and we

now have the tilting furnace in common use. This furnace is fired either by gas or gasoline; it rests on trunnions, and can be tilted through more than 90° by means of a crank, thus discharging the contents of the crucible, and relieving the melter from exposure to the heat. The burner is fastened rigidly to the pipe-line and must be shut off before the furnace is tilted. I am using a No. 40 Donaldson tilting furnace manufactured by the Denver Fire Clay Co., and have modified the burner connection at an expense of about $3, thus greatly increasing the usefulness of the furnace. The picture shows the furnace tilted ready for cupelling, with the burner made fast to the furnace by means of strap-iron, and connected to the gasoline pipe-line by a flexible copper hose furnished by the Pennsylvania Flexible Metallic Tubing Co. The copper tube is nearly as flexible as the ordinary steel hoisting cable of the same size, and



TILTING MELTING FURNACE

is guaranteed to be non-corrosive and to stand 100 lb. pressure. With the burner connected in this way the furnace can be tipped to any angle below the horizontal without interrupting the blast. I have cupelled in this furnace, forming liquid litharge, which was drawn off through a pipe connected to an air-line by means of a hose. For this work a cement cupel was built in an old crucible. A board was made to fit snugly into the crucible, dividing it, in the direction of the axis, into two equal parts and to the under side was fastened the pattern which was to shape the cupel bowl. The cement was then tamped into place and the board slipped out and the pattern removed. The pattern was 7 in. long, 6 in. wide, and 1¾ in. deep, with sloping sides, and gave a bowl which after making allowance for litharge gutter had a capacity of 25 lb. of lead.

This furnace can also be used to demonstrate the roast-reaction process of lead smelting. The hearth bottom is built in the crucible,

as previously described, except that the pattern is left out, thus giving a hearth with a plane surface in which a lead gutter can be cut. The pipe for applying the air-blast should be straight, and direct the blast against the side and end of the crucible, thus allowing the air to reverbrate and become preheated before it comes into contact with the material to be roasted.

The accompanying table, showing a temperature test on this furnace, may be of some value to those considering the use of this type. The gasoline used was 62°B. (really petroleum naphtha) and was consumed at the rate of one gallon per hour. The temperature determinations were made with a Bristol thermo-electric pyrometer, with the fire-end placed about one inch from the bottom of the crucible. At 11 o'clock, after a preliminary warming and drying for one hour, the blast was turned on full force. At this time the outside of the furnace was barely warm. The blast was turned off at 11:35, and the continued rise in temperature indicated at 11:37 was probably due to a lag in the pyrometer.

Time of day	Degrees F.	Change
11:00	1557	..
11:05	1600	43
:10	1680	80
:15	1744	64
:20	1800	56
:25	1850	50
:30	1890	40
:35	1920	30
:37	1930	10
:40	1890	40
:45	1820	70
:50	1750	70
:55	1680	70
12:00	1590	90
12:05	1530	60
:10	1465	65
:15	1390	75
:20	1330	60

CONVENIENT SLAG FURNACE

By J. D. Hubbard

(April 27, 1912)

A description of this slag furnace and the method of reduction of slags obtained from the melting of cyanide precipitate at the Oriental Consolidated Mining Co.'s plant at Taracol, Korea, were given in the February 5, 1910, issue of the *Mining and Scientific Press*. A further description and details of construction, may be of interest to the profession and assist some one who is confronted with the same problem. In the preceding article the cost per ton of slag treated was given at $32.89. This was no doubt a shock to the economical, and prevented any further consideration of the furnace, as many could ship their slags to the smelter for a less cost per ton than $32.89. In actual practice with the furnace it

has proved conclusively that the costs were much less. The elimination of coke for fuel (which costs $30 per ton at Taracol) and the substitution of charcoal, also the reduced labor and other costs, have resulted in lowering the net cost to $7.51 per ton of slag treated, as follows:

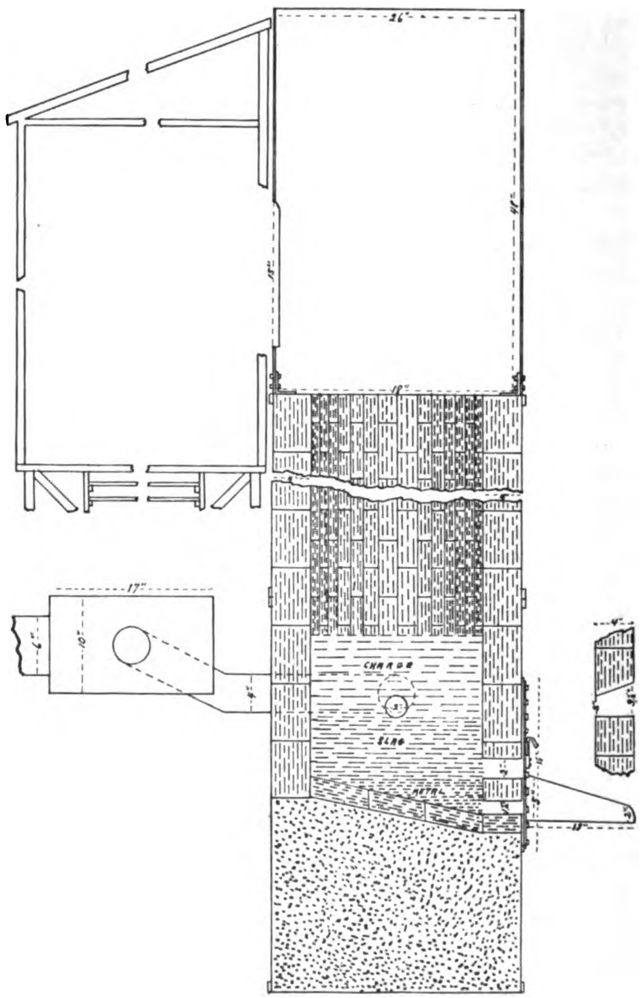
Charcoal	$4.50
Labor	1.66
Power	0.17
Iron, etc.	0.12
Firebrick	0.84
Depreciation (25%)	0.22
Total	$7.51

This cost is less than the treatment charge at the smelter, to say nothing of freight. Adding the advantages of local treatment products where posible, and the high net extraction obtained on a refractory slag, this slag furnace offers advantages.

Of course, the labor item would be cheaper in the Orient or other low-priced labor countries than in the United States; but, on the other hand, fuel, especially where oil can be used, would be much cheaper than in the States. One-half ton of coke is amply sufficient to reduce one ton of the most refractory slag. Where the slags contain only shot gold or silver, re-grinding in a clean-up barrel with quicksilver would practically recover all the precious metals, but in most cases where they are held in matte, drippings from crucibles, scales from the roasting-pan, etc., from a low-grade bullion product, it necessarily follows that a different metallurgical process must be employed. The cleaning property of lead on a charge is too well known to require reiteration. It follows that if the proper amount of lead be added to a slag containing gold and silver in a metallic form, a satisfactory elimination of gold and silver will result. Acting upon this principle, this slag furnace was evolved and perfected as far as possible. It has proved quite satisfactory in Korea and has saved the Oriental Consolidated Mines Co., where it was developed and put in use, much money.

The plan of the furnace is simple, as can be seen by the drawing, and any small blacksmith shop would be equal to the construction of one. The furnace shown in the drawing is large enough to treat two tons of slag in ten hours. But the capacity can easily be increased by proportionately increasing the dimensions of the furnace. The shell may be made of any old sheet iron available, as its use is merely to hold the inside lining in shape. The shell will last indefinitely if the lining is kept intact. It is always necessary to renew a portion of the lining after every heat. At first firebrick alone was used for the lining, but I found that a mixture of red clay, broken fragments of graphite crucibles, and discarded firebrick, all broken to pass a 1-in. ring, was just as effective for lining, and of course much cheaper. The only firebrick used are to line draft-holes and tap.

Another point of economy was that it was not necessary to buy a single pound of lead for use in the furnace, as slag from the assay office, which always contains unreduced litharge as well as borax and sodium bicarbonate, both useful in liquefying slags, was



CONVENIENT SLAG FURNACE

used for the purpose. The charcoal used as fuel brought all the lead down in good shape.

The method of treatment was fully described in the article mentioned. Briefly, it is to mix the slag with the required amount of fuel and assay slag, and feed it in slowly at the top of the furnace. A proper amount of forced draft must be maintained, and care

taken not to overfeed the furnace. A roof over the feed platform is quite desirable for protection from red-hot cinders and particles of melted slag. It is essential to keep the draft-holt in good shape by reducing the diameter of the inside end of the opening, as shown in the drawing. The sharp angle formed by the top brick prevents the slag from backing up in the draft-hole. Also the hinged door at the tap or some other arrangement for opening and removing the residue of the charge at the end of the heat is necessary, so as to leave a clean furnace for the next charge.

The actual extraction by bullion returns, and assaying the final slag at Taracol, Korea, was 98% on over $40,000 worth of slag. The bullion produced is necessarily base, averaging at Taracol 34.45 gold and 259.51 silver. There is enough excess lead in the base bullion for satisfactory cupellation, yielding clean doré bullion. A shallow cupel, 3 in. deep in the center and 10 in. wide across the top, is a satisfactory size to use, and does not require a very large cupelling furnace. The cupelling furnace may be of simple construction, care being taken, however, to have it sufficiently large on the inside to allow plenty of fuel around the muffle. The muffle should be large enough to admit two cupels 3 by 10 in. Each cupel will hold 25 lb. of bullion. A patent has been applied for on this furnace to protect it from speculators, but it is freely offered to metallurgists and mining companies who may find it useful. My thanks are due to W. H. Aldridge, master mechanic, and B. Pedersen, his asistant, for valuable hints given in the construction of this furnace.

TREATMENT OF MATTE FROM MILL CLEAN-UP

By M. W. VON BERNEWITZ

(March 2, 1912)

Matte is a by-product from the smelting of gold precipitate from cyanide plants, and the subsequent refining of the bullion by sulphur. Its formation cannot be prevented; it carries high gold content, is difficult to treat, and is generally a nuisance.

There are many methods of treating matte; among others being the roasting and smelting, acid treatment, and the cyanide reduction. On the whole, results have not been too satisfactory. I recently made an atack on an accumulation of matte at the Associated Northern Blocks. This material was produced from the cyanide bullion by refining with sulphur. It is high in Cu, and contains Fe, Zn, and Pb. It is melted in a No. 50 crucible, and lumps of sulphur are kept under the surface by means of a small inverted clay pot held by tongs. The contents are poured into a conical mould, the matte broken off the bullion as clean as possible, and the latter finally remelted. It may be mentioned here that the matte sticks to the bullion and has almost to be chopped off.

In reducing the matte I used the cyanide method, as described by A. E. Drucker in the *Mining and Scientific Press* of May 18, 1907. Briefly, this consists of crushing the ingredients to pass a

4 by 4 screen, and putting layers of borax, matte, and cyanide in a No. 60 crucible, and so on until the parcel is through. The matte and cyanide was crushed by pestle and mortar. Some 50% by weight of cyanide and borax was used, less flux and reducer give low results.

The crucible when hot was charged with layers of each material. This did not take long to run down and a thick scum or dross filled the pot. This was skimmed off by means of a half-round skimmer, about 6 by 2 in. at the widest part. More matte and flux was added during a three days' campaign of some 6 hours each. During the melting a good deal of fume was given off and it was necessary to be careful. At the end of each melt the crucible was poured into a conical mold, and a most noticeable point was that some unreduced matte did not stick to the bullion at all, this being due to the cyanide flux. I would therefore recommend the addition of a little cyanide before pouring any melt that contained matte, preferably mixing it a trifle. Following are some details of the treatment:

	Pounds
Weight of matte	145
Weight of cyanide used	72
Weight of borax used	72
	144
Bullion recovered	30.5

	Au per ton, ounces.
The first matte assayed	4640
The second matte assayed	500
The slag, dross, or residue	54

This shows a high concentration and extraction. The second matte and dross was bulky and difficult to treat; it was therefore fed into the ball-mills during the next two days and treated along with sulphide ore. The gold in this stuff must be in an extremely fine state, as it could not be detected in any samples. There was only about 40 lb. of the second matte. Sulphur did not act readily on the final bullion from the matte, so it was poured and assayed. The fineness was 540 in gold and 220 in silver, base being 240. The cost of the above treatment was as under:

Wages	$ 9.70
Fuel	4.85
Cyanide	15.15
Borax	7.49
Sundry	1.56
	$38.75

Not wishing to make more rich matte by further sulphur refining, and not liking to use nitre, we decided to try bessemerizing the bullion. A temporary pipe-line from the air-compressor to the furnace was made out of ⅜-in. pipe and ½-in. rubber hose. It was

necessary to take the air from this point, as it was warm, averaging 91°F. at the furnace. For conducting the air into the molten bullion, we used the long clay 'Churchwarden' pipe stem, the rubber hose being bound on with asbestos twine. The stems were gently heated, and lowered into the bullion after air at 90-lb. pressure was turned on. It takes very little air indeed to keep 400 oz. of bullion well agitated. A little borax flux was used, and this picked up the base as it rose to the surface. Fairly dense zinc fume went off for some time; the air did not in any way cool the bullion; but quite the reverse, the whole mass seemed to be very hot and active. After an hour's air treatment the pipe stems got too short to be easily handled, so the bullion was skimmed with sand, and poured. The final bar was 590 fine in gold, 210 in silver, and 200 in base, showing that, with a prolonged bessemerizing of, say, 3 hours, with decent apparatus, bullion 900 or more fine could be produced. So, along with the matte treatment described, I would recommend bessemerizing low-grade bullion, as it is a cheap and easily worked method. Although not really conclusive, the bessemerizing shows that the bullion can be refined in this way, as described by Rose. Further experiments with proper apparatus will be tried.

(April 6, 1912)

The Editor:

Sir—I have read with considerable interest the article by M. W. von Bernewitz in the *Mining and Scientific Press* of March 2, I also tried the process of treating matte by reducing it with potassium cyanide soon after reading A. E. Drucker's article, but I did not like the process, as it yielded a thick pasty slag, rich in gold and silver. I am operating a cyanide plant and treating about 80 tons per day of mill tailing and canvas sweeps. I dip my zinc-shavings in a solution of lead acetate when charging them into the precipitating-boxes, hence my melt contains quite a quantity of lead in addition to the usual matte. My process of treatment is as follows: I clean up twice a month and the acid-treated precipitate is melted in two furnaces, using No. 25 plumbago crucibles. The result is about 15 lb. of lead, and matte, in addition to the precious metals. This I oxidize and refine in about three hours' time, by using a modification of what Mr. von Bernewitz terms bessemerizing; but which I think might more properly be called crucible cupellation.

I have a ⅜-in. air-line from the air-compressor, ending with a valve to which is attached about 6 ft. of ½-in. rubber hose, and 4 ft. of ¼-in. pipe for a nozzle. The pipe-line runs along the roof of the melting-room, so that the hose and nozzle come down from above, and are made adjustable by means of a rope that is fastened to the air hose and to one of the rafters. The melting pot is covered with a plumbago cover, having a small slot in one side to admit the end of the pipe, which is adjusted so as to enter the melting pot at an angle and is kept about 4 in. above the surface of the melt. I use a large quantity of air under about 10 lb. pres-

sure, and add borax and silica to absorb part of the metallic oxides and protect the pot. The melt gives off dense fumes of zinc, lead, and sulphur, so that it is impossible to remain in the room even with the windows open in addition to top ventilation of the building. I shut the air off and skim off the slag when it gets thick enough to interfere with the cupellation of the base metals. When the matte and lead are oxidized the dense fumes cease and the melt is poured. This gives a bar about 550 fine in gold and 250 in silver, the remaining base being chiefly copper. This can also be removed by long-continued blowing and fluxing, but is unprofitable.

I once made a lot of clay pipes about 3 ft. long and tried refining by Rose's method of blowing air into the metal, but it bubbles violently and spatters metal all over the room if a strong blast is used, while if only a small quantity of air is admitted it works much slower than when using a strong blast on the surface of the melt. When refining I keep the furnace covered, with the exception of a small opening to admit the ¼-in. pipe into the furnace, and through the hole in the crucible cover, and maintain a high temperature, as the matte will not burn at a low one.

WILTER E. DARROW.

Amador City, California, March 8.

(July 13, 1912)

The Editor:

Sir—In your issue of March 2, M. W. von Bernewitz has an interesting article on the treatment of matte and sulphur refining, but I am by no means convinced that the method of smelting the matte with cyanide and borax is the most convenient or best. Lately I have not had any matte to handle, having come to the same conclusion as Mr. von Bernewitz, that it is a nuisance and that its formation is best avoided, but, some time since, I did a good deal of sulphur refining, of which the following are typical examples: Crude bullion from cyanide clean-up, weight 171.7 oz. Assay value: gold, 518.5, equivalent to 89.03 oz.; silver, 264.2, equivalent to 45.37 oz.; base, 217.2, equivalent to 37.30 oz. This bullion was now melted, keeping the heat as low as possible, in a 30 'salamander' crucible, and 15% of its weight of sulphur added, stirring with a salamander rod. The large crucible was used to give plenty of surface and to avoid loss through spitting. The use of a small clay crucible inverted over the sulphur, described by Mr. von Bernewitz had been previously tried and abandoned, as it was thought the action was too violent and that some gold was lost by spitting.

The resulting bullion weighed 92 oz., assaying: gold, 813.9, equivalent to 74.878 oz.; silver, 139, equivalent to 12.788 oz.; base, 47, equivalent to 4.324 oz. The resulting matte, weighing 93.8 oz., was smelted in a hot fire with iron, no other addition being made except a little borax. The resulting bullion weighed 49.4 oz., assaying: gold, 301.4, equivalent to 14.8 oz.; silver, 548.2, equivalent to 26.9 oz.; base, 150.3, equivalent to 7.7 oz. The resulting matte

	Bullion, oz.	Au, %	Ag, %	Base, %	Au, %	Ag, %	Base, %	S added. oz.	S added. %
Crude bullion	171.50	51.85	26.42	21.72	89.03	45.37	37.30	24	15
After first S treatment........	92.00	81.39	13.90	4.70	74.87	12.78	4.35	..	..
Bullion from matte............	49.40	30.14	54.82	15.03	14.80	26.90	7.70	..	..
Residual matte contains	......	0.05	10.00	Cu 35.00	0.026	5.18	Cu 18.13	..	..
Crude bullion	238.00	46.13	35.32	18.34	109.90	84.50	43.60	24	10
After first S treatment........	183.75	59.89	31.82	8.28	110.04	58.47	15.24	24	13
After second S treatment......	150.75	70.49	25.68	3.82	106.26	38.73	5.76	7	5
After third S treatment........	141.50	73.54	23.94	2.51	104.06	33.78	3.56	..	..
Bullion from matte............	51.30	13.24	72.68	14.07	6.79	37.22	7.29	..	..
Residual matte contains.......	......	0.027	13.70	Cu 33.50	0.025	12.35	Cu 30.15	..	..
Crude bullion	105.00	49.10	39.70	11.20	51.56	41.69	11.75	16	15
After first S treatment........	65.50	71.22	25.60	3.17	46.65	16.77	2.07	..	..
Bullion from matte............	27.25	17.96	72.08	9.95	4.89	19.64	2.71	..	..
Residual matte contains.......	......	0.05	12.18	.....	0.021	5.28	.....	..	..

RESULTS OF TREATMENT OF MATTE BY SULPHUR.

weighed 51.8 oz., assaying: gold, 16.45, equivalent to 0.52 dwt.; silver, 10%, equivalent to 5.18 oz.; copper, 35%, equivalent to 18.13 oz. The matte was not treated further, but shipped to the smelter with the slag. It will be seen that the bullion assays do not agree exactly, owing to the difficulty of getting a fair average sample of the low-grade bullion by boring; it should have been sampled by dip samples.

The net result of the sulphur refining was to reduce the base metal from 37.3 to 12 oz. and to lock up 5½ oz. silver and ½ dwt. gold in the matte. The following is a summary of this and of two other lots of bullion treated in a similar manner.

This shows that the 'iron' method of treating matte is effective in recovering gold, but the resulting bullion is too base, requiring further refining, and a notable quantity of silver is locked up in the matte. It seems to be more effective than the cyanide method and is certainly easier, no pulverizing of the matte being required, and cheaper, no cyanide and but a very little borax being needed.

Not being satisfied with the sulphur treatment, I thought that if the precipitate were slightly roasted before acid treatment, in order to oxidize and render more soluble the base metals, a residue would be left containing little base metal, and smelting charge would be reduced. This was found to be the case, with the additional advantage that there was no foaming, as no hydrogen was evolved, less acid was required, as part of the zinc was volatilized, and any white precipitate was decomposed by the roast and rendered more soluble. On the other hand, there was the possibility of loss of precious metals in roasting. I was skeptical as to there being any serious losses, but to determine the matter to my own satisfaction, I made the following experiments. A difficulty in carrying out work of this class arises from sampling, it being almost impossible to get accurate samples of such rich stuff; however, all reasonable precautions were taken and six determinations were made to reduce the chance of error as far as possible.

After roasting or treating with acid, the samples of precipitate were fluxed as follows: precipitate, 100 parts; litharge, 50 parts; borax, 50 parts; soda, 25 parts. The slag was then cleaned by smelting with more litharge and iron, and the lead bullion cupelled, and assayed. The precipitate experimented on was from the treatment of an old slime dump and was essentially low grade.

EXPERIMENTS ON SHORT ZINC

		Au, gm.	Au, gm.
1*a*	500 gm. ppt. after roasting yielded	11.84	10.51
1*b*	" " " acid treatment	12.14	11.06
2*a*	" " " roasting	7.77	7.26
2*b*	" " " acid treatment	9.69	8.22
3*a*	" " " roasting	6.92	5.95
3*b*	" " " acid treatment	8.43	7.02

Average loss by roasting: gold, 12.3%; silver, 9.9%.

The apparently smaller loss of silver was probably due to some being dissolved by the sulphuric acid.

EXPERIMENTS ON FINE PRECIPITATE

		Au, gm.	Au, gm.
4*a*	500 gm. after roasting yielded	15.10	13.02
4*b*	" " acid treatment	13.97	14.59
5*a*	" " roasting	10.84	10.52
5*b*	" " acid treatment	11.08	10.53
6*a*	1000 gm. " roasting	40.02	37.94
6*b*	" " treatment with lead acetate to replace zinc and smelted without either acid treatment or roasting...	43.14	42.50

Average loss of gold by roasting, 3.3%; silver, 8.8%.

Here the loss of gold by roasting is much reduced, but the silver loss is relatively higher, probably owing to No. 6*b* not being acid treated. In the six parcels treated, five gave higher results without roasting, and in the case of the sixth (No. 4*a*) the combined weight of gold and silver was higher in the non-roasted portion, and it is possible that an error occurred in assaying the bullion, parting may have been imperfect. Roasting was effected in an iron pan over a fire; possibly with a more perfect roasting arrangement the losses would be materially reduced.

It seems probable that nearly all the advantages of roasting before acid treatment could be obtained by subjecting the damp precipitate to treatment in a chamber with warm air saturated with moisture and a little acid vapor (acetic acid would probably be suitable), somewhat on the lines of the old Dutch process of white lead manufacture. All the metallic zinc and copper would be readily oxidized and rendered soluble in dilute sulphuric acid, but it is doubtful whether the expense of this preliminary treatment would in many. cases be justified.

The dried precipitate, after acid treatment, is in a finely divided state, and is specially suitable for smelting with an oxidizing flux. My present practice is to smelt with MnO_2, borax, sand, and flourspar, obtaining a bullion that averages about 3% base metal. The silver is to a considerable extent carried into the slag, but that is locally no disadvantage, as the mint allows only 1s. 9d. per ounce for fine silver, while the combined bank and mint charges are 1s. 2d. per ounce, and as the slag has in any case to be shipped to the smelter, it is more profitable that it should have a high silver content. MnO_2 acts more energetically than nitre, and does not boil up so much, nor does it seem so corrosive on crucibles. Bessemerizing by air was first tried and patented by Manton, of Menzies, but the corrosion both of the air-pipes and of the crucibles was excessive, and until Rose published his classic paper on this method, no really conclusive work had been done to prove that there was no appreciable loss of precious metals.

H. R. EDMANDS.

Menzies, Western Australia, May 10.

NEW METHOD OF ZINC PRECIPITATION

(March 15, 1913)

JOHANNESBURG CORRESPONDENCE

In a paper just read by John S. MacArthur before the Chemical and Metallurgical Society of South Africa, the use of zinc wafers in preference to any other form for precipitation purposes was recommended. It was pointed out that all attempts to displace zinc by the use of charcoal or electricity had failed, and as an effective chemical agent, filiform zinc had shown itself hard to beat, although C. W. Merrill had successfully introduced the use of zinc dust.

Probably the high chemical activity of the zinc thread had much to do with its success, for if it is examined under a lens it will be seen that while one side is polished by the cutting tool, the other side has a velvety pile exposing a large surface to the auriferous solution, which accounts for the prompt responsive action of this form of zinc. The velvety pile seems to encourage the liberation of free hydrogen accompanying the precipitation of gold. The disadvantages of filiform zinc are due to its structural and mechanical considerations involved in its use and in dealing with the bullion precipitate. Then again, zinc thread is weak and its fibrous structure is destroyed by the cutting tool, the velvety surface being only an indication of the thread having been strained beyond the breaking point of the original fibre. The solution soon penetrates the pile of velvet and the spongy zinc thread collapses into a dirty mush before its full chemical work is accomplished. Thus it is that bullion precipitates generally contain at least 50% of disintegrated zinc needing elaborate methods of refining, which, being complicated by the dusty nature of the precipitate, call for stringent precautions against loss. There is another disadvantage attached to the use of zinc shaving, and that is the difficulty of packing it with perfect regularity, some parts being tight and others comparatively loose, resulting in the solution selecting the easiest path where the possibilities of contact and precipitation are least. This is contrary to what we wish, as it results in only a small proportion of the solution passing through the tight part where the possibilities are, of course, greatest for the consumption of zinc chemically, and the disintegration is just about the same, whether the precipitation is active or sluggish.

Mr. MacArthur had aimed to overcome these drawbacks by using zinc sheet cut into wafers of convenient size, or to two inches long by one-quarter to one-half inch wide. Oblong wafers pack better and are more conveniently handled than when square. The wafers are made by cutting ordinary sheet zinc of a convenient gauge, say No. 11, into 1½-in. strips in a bookbinder's guillotine and then cutting these strips crosswise so as to produce wafers approximately 1½ in. long by ¼ in. wide. When these wafers fall into the cell of the zinc box they arrange themselves irregularly, more or less overlapping in slate fashion. The corners of each

wafer are somewhat distorted by the guillotine, which prevents any sticking, providing at the same time channels for the uniform passage of the auriferous cyanide solution, ensuring an equal opportunity of precipitation in all parts of the mass.

The resistance to the flow of the solution is much greater with wafers than with filiform zinc, making it necessary for the extractor boxes to have at least twice the usual fall, while the cells and the boxes may be made with only half the usual depth. This method of precipitation was introduced at the Caveira mine in Portugal, in 1907, where the ore contains about 2.5 gm. of gold and 120 gm. of silver per ton. With coarse-crushing the average extraction is about 80% of the gold and 75% of the silver. The cyanide plant was equipped with the usual clean-up plant, but as the bullion precipitate only contained 3% of zinc and 12% of other foreign matter it was found that no acid treatment was necessary. Careful observations show that in use the zinc wafers become thinner and thinner until at the end of three weeks or so they disappear altogether without the structural strength being impaired or leaving distinguishable débris. This new method of treatment has proved itself economical in the consumption of zinc, has greatly simplified the clean-up, and thus saved time and temper.

The use of this kind of zinc has been described by Messrs. Lloyd and Rand in connection with a rotary extractor, in 1909. The operations at the Caveira mine some two years earlier showed that a revolving extractor-box was unnecessary in using zinc wafers, and as the method has been tried both in Italy and Mexico with equally good results, its success is now proved. It may be said that at the Caveira mine in Portugal the precipitation is facilitated by the presence of a large amount of silver with the gold, but in Italy the mine produces only a low-grade gold ore without silver, and as the wafers do their work well there also, it may be assumed that they will answer anywhere. The cost of installing the guillotine, the only special appliance needed, is only £10 or so, a much less sum than that required for a zinc turning lathe of equal capacity.

DISPOSAL OF RESIDUE

DISPOSAL OF TAILING AT EL TIGRE, SONORA

By Donald F. Irvin

(December 7, 1912)

The cyanide residue discharged from the plant of the Tigre M. Co. is an all-slime product, about 85% of which will pass a 200-mesh screen. Of the ore before grinding, about 60% is a natural clay slime, the rest requiring re-grinding in tube-mills. This product is handled by Kelly filter-presses, and from them flows to waste.

In common with many mining properties operating cyanide plants in Mexico, the Tigre M. Co. has been obliged to prevent the injury of neighboring stockmen and ranchers by the poisonous effect of cyanide solution in the tailing discharged from its reduction plant. The stockmen are quick to resent such injuries, and in consequence cyanide residues must be judiciously handled, even in districts that are practically a wilderness, if lawsuits and injunctions against operation are to be avoided, to say nothing of bills for cattle poisoned by creek water. It should also be recognized that the Mexican bench, in decisions rendered in suits tried in southern Mexico, has affirmed the prior right of the agriculturist as opposed to the miner, so the rancher has the weight of the law with him in his contentions.

When cyaniding was begun at El Tigre this slime product was discharged directly into Tigre canyon, which is the bed of a torrential mountain stream, in the belief that the natural decomposition of the cyanide contents would be effected before reaching the Bavispe river, some 15 miles distant. The latter stream, during most of the year, carries a large volume of water, and supplies the municipality of Oputo, an irrigated district, some 60 miles south, and other water users as well. A very short run proved that there was poisoning of local livestock in Tigre canyon, so the stream-bed was immediately fenced with barbed-wire. This was possible, as the Tigre M. Co. owns all the land intervening, although it is used as a cattle range by various Mexican and American stockmen on both sides of the canyon. As a preventive, it served its purpose against cattle poisoning in the canyon, but the question of the quantity of cyanide in solution at the junction of the Tigre canyon and the Bavispe river was vigorously raised by the people of Oputo. In their minds, their case was strengthened by an epidemic of typhoid fever, which incited the management of the Tigre M. Co. to further measures designed to absolutely remove all chance of pollution of the canyon water, and coincidentally avoid any legal clash with the irrigation districts on the Bavispe river, although there never was a trace of cyanide found in the latter stream at any time. Solution samples taken in the Tigre canyon 50 ft. from its junction with the Bavispe failed to show a trace of cyanide.

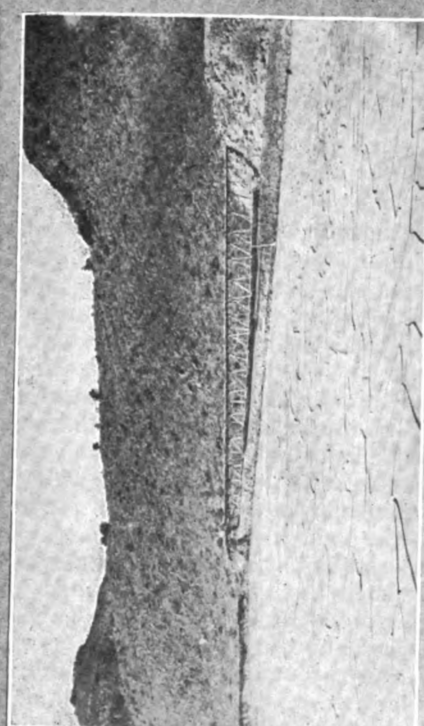
Various plans for storing the cyanide residue were considered, as that was decided to be the only desirable plan. Discharging the tailing dry and stacking with a belt-conveyor, in the manner used by the Zambona M. Co. in Sonora, was not feasible, as there is but

very little storage room close to the filter plant. Damming the canyon itself is a dubious expedient, as it has a heavy grade and is not wide; furthermore, the summer rains are most violent, and at times approach cloudbursts in their intensity. Hence it was decided to retain the slime upon a high mesa, about two miles down the canyon from the cyanide plant. This mesa lies some 150 ft. above the stream-bed, and is about 40 acres in extent, about one-half of which is adapted as a unit for a reservoir site.

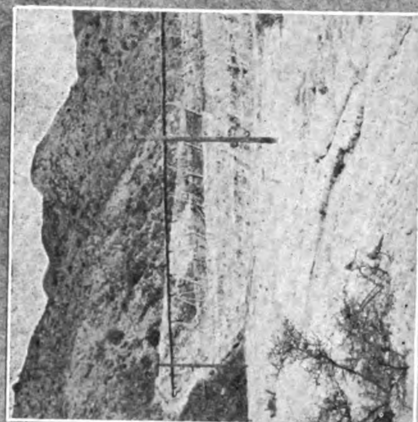
Knowing from previous experience elsewhere, as at Goldfield, that slime dams may be built up from their own semi-dried mud, used as wall material, the work upon the dam was started in the winter of 1911-1912, and also upon excavation of a ditch along the canyon wall from the filter plant to carry the slime to the damsite. The first work was the throwing up of a low preliminary earth and rock dam to hold the first of the slime, until it became dry enough to be useful as wall material. A low ridge in the centre of the reservoir site, running at right angles to the face of the dam, divides the storage room into two parts, and it was used from the beginning as a means of dividing the pond into two basins, one receiving the flow of pulp while the other is thickening.

Inasmuch as settling recovers a fair amount of solution, in spite of loss by evaporation, and this solution contained small quantities of precious metals, cyanide, and lime, a decanting pipe was placed in each bay of the reservoir and the decanted solution was run through a zinc-box connecting with a triplex Aldrich pump, which can either return the solution to the mill or through a by-pass into the slime ditch above the point where it enters the dam. The pump was installed as a means of recovering all the water possible in the dry season, as well as for the sake of the solution itself. However, the mill is now running on a much smaller water consumption than was at first thought possible, and, as the natural water-supply is more abundant than formerly, the pump is used only as an adjunct to natural agitation of the thick slime tailing in the ditch as it comes from the filter plant to the reservoir. By dilution it thus effects a slight additional recovery of dissolved metal, which is caught in the zinc-box, although, as a rule, this is so small that the recovery thus made is quite insignificant. The dissolved metal in the pulp is now much less than was thought at first could ever be attained, hence the recovery in the zinc-box is small. The amount of solution decanted from the surface of the reservoir will vary with the seasons, naturally being greater in the rainy months —a low figure being 3 gal. per minute and a high being 12 gal. per minute. Percolation is now negligible, the other sources of loss of water being from evaporation and retention by the slime itself.

The essential benefit of the arrangement is that, beyond dispute, all cyanide tailing is kept from polluting the natural waters of the district, and the tailing is preserved on ground belonging to the Tigre M. Co. Although at present further metal recovery is not practicable by current metallurgical methods (they assay only 1.25 to 1.50 oz. silver per ton), at some future time they



TAILING POND



TRESTLE FLUME

may represent a live asset. The discharged residue from the filter plant is very thick, and unless diluted by a small stream of water that enters the launder would not flow in the ditch, in spite of its heavy grade, which is 1 in 9, or about 11%. A series of determinations over a considerable period of time on the pulp issuing from the filter plant show an average moisture ratio of ½ to 1.

Before the additional water was added to the slime ditch the pulp would often back up in the ditch and overflow at the turns, and at every culvert that crossed the small side *arroyos* the slime would leap over the slime of the launders. Raising the dam walls, cars of the ditch and of the pulp involve some labor, and for this purpose three Mexican laborers live at the damsite and do all this routine work. These men are paid ₱2.50 and ₱2.25, and are the only labor charges incurred in maintaining the work. The carpenter work required in building the distributing launder along the crest of the dam was, of course, charged to construction.

In operating the dam it is thought best to discharge the slime as near the dam itself as possible, so that the heavier deposit may form there and the thinner part gravitate to the back of the pond, where the decanting pipes are placed. The photographs show the arrangement of discharge of the slime into the dam through a trestle flume, and also the manner in which the dam is built up. The trestled flume was chosen as a means of conveying the sluggish pulp along the crest of the dam and discharging it at that point with the minimum of labor and time lost in alterations, as the level of the slime pond rises.

Seeking added stability for the dam, an auxiliary dam was started below the main dam, after impounding of tailing had begun. This was commenced in much the same manner as the main dam was built, save that the slime was run into the second catch-basin through holes made in the wall of the big dam and allowed to flow in a series of small sections.

This lower dam is 50 to 75 ft. from the face of the main dam and is kept about 7 ft. lower. The crest of the main dam is 2 to 3 ft. thick and is seldom more than 1 or 2 ft. above the surface of the pond. The thickness of that portion of the wall that is below the surface of the slime in the pond is indefinite, for it is homogeneous with the slime of the pond itself, and hence it may be considered to be any thickness. This was shown by a washout during the past rainy season, when a crevasse was formed in the face of the dam during a tremendous downpour. The part washed out was narrow and deep, and showed the slime to be uniform, leathery, and compact from top to bottom, and this condition persisted for a number of feet back from the face of the dam. At the present writing (about November 1, 1912) the main dam holds, in round numbers, 60,000 to 65,000 tons of tailing and the crest of the dam is about 20 ft. above the ground-level at the highest point.

This method of tailing storage has no essential feature of special novelty, although it is unlikely that such a long and devious

ditch is in use elsewhere to convey the tailing, and the remarkably coherent Tigre slime enables the dam to be built to a height that would otherwise be unsafe. In this connection it may be added that the face of the dam may be built up so as to impound tailing for at least three years at their present rate of increment, before it will be necessary to begin the construction of a second dam on the more roomy site immediately below the present one.

CONCENTRATION OF DISSOLVED METALS IN SLIME PONDS

By M. W. VON BERNEWITZ

(January 18, 1913)

A millman at Tonopah recently said to me, "When anybody states that there is no dissolved metal going out with the residue, I doubt it." And most people will agree with him. At Kalgoorlie, especially from the ponds consisting of finely ground roasted ore, the clear water floating off has at times assayed from 6 to 20c. per ton, and has been clarified, strengthened by addition of a little lump cyanide, and the metal content precipitated on zinc shavings. The water always shows cyanide by test, at one mine averaging 0.003% KCN. At another mine, lessees of the ponds did fairly well for a time; but on the whole there is little profit to be made in this way.

In *The Mining Magazine* for June, 1910, G. B. Butterworth describes the sampling, erection of plant, and results of a dump of roasted ore which had been treated in the usual way at Kalgoorlie and discharged from cars. In this case, sampling was inaccurate, as "this is a case in which a portion of the gold in the residue was converted into soluble gold, in the presence of even such a small quantity of cyanide as remained after the final fresh-water washing, during the air-drying and subsequent handling, until it was buried in the dump. The rain soaking through would then be a solvent, and the sun would draw the moisture, carrying gold, to the outside crust; this would be deposited by evaporation as a soluble salt of gold."

In the Tonopah district quite a new industry has sprung up of late in recovering from the slime ponds the soluble silver salts which rise to the surface by the combined action of moisture in the dumps, heat of the sun, and capillary attraction, or in other words, evaporation of weak cyanide solution which carries dissolved metal. The salts form a crust with the top ⅛ in. of slime, and are collected by scraping with garden hoes or whisk-brooms. A great deal depends on the weather, and warm weather is most desirable. The value of the product depends on the care taken by the men in scraping it up, as digging too deep is like adding waste to ore. I was told that the *caliche,* as it may be called, varies in value from $30 to $130 per ton,

and will average $40 per ton, while the cost of scraping and treatment also varies, since one man can collect 2000 lb. per day at $4, if the deposit is thick; but he will average only about 1000 lb. with the usual thin layer. Then there is the company's royalty to be paid which amounts to 35% in one instance. At the Belmont mill, at Millers, the *caliche* is sampled, fed into a tube-mill, and then mixed with the pulp, undergoing the usual treatment, while at the Extension it simply passes through with the ore after being sampled. One lessee ships his product to the Selby smelter at San Francisco. Even if the cost of collecting and treatment reaches $13 per ton (a figure given me as correct), there is a good profit in it for both the lessees and the companies.

TRANSPORTATION OF TAILING THROUGH PIPES

(April 6, 1912)

The Editor:

Sir—I have been much interested in several articles which have appeared in your journal concerning the transportation of battery pulp through iron pipes for great distances, and as data on this subject are rather scarce, probably the enclosed item (see table p. 441), which was very kindly sent to me by T. J. Grier, superintendent of the Homestake Mining Co., at Lead, South Dakota, will prove of general interest to your readers.

The data are particularly interesting on account of the fact that the experience covers a period of eleven years.

S. B. CHRISTY.

Berkeley, California, March 20.

HOMESTAKE MINING CO. DATA OF FLOW OF TAILING IN PIPE

	Lead Sand Line.	Central City Tailing Line.	Lead Slime Line.
Diameter of pipe	12 in.	8 in.	12 in.
Thickness of pipe	9/16 in.	9/16 in.	9/16 in.
Average grade	Part at 5% Part at 2½%	1¾%	1½%
Bends in line	The pipes generally follow the contours of the hills; the sharpest curve is 22½°, a standard elbow for flanged C.I. pipe being used.		
Fineness of tailing	30% on 100-mesh 25% on 200-mesh 45% through 200-mesh	25% on 100-mesh 20% on 200-mesh 55% through 200-mesh	100% through 200-mesh
Water in pulp	70 to 75%	85 to 90%	65 to 70%
Solids per 24 hours	1950 tons	450 tons	1100 tons
Pipe runs	About 1/3 full	Full	
Wear of line	(It is customary to turn pipe through about 120° when bottom is worn thin; thus the pipe will be used until three lines of wear have been developed.)		
	On 5% grade, worn out in 2 years. On 2½% grade, worn from 9 to 10 yr.; most of line still in use after 11 yr., but badly worn.	In use 8 yr. without turning.	In use 5 yr. without turning.
Remarks	Grade has been reduced to 2½% throughout, and new pipe 1 in. thick recently installed, for use in another month.		

MEASUREMENT AND ESTIMATION OF TONNAGES

CAPACITY OF CIRCULAR VATS PER FOOT OF DEPTH

By W. A. CALDECOTT*

(September 24, 1910)

1 TON = 2,000 LBS. = 32 CUB. FT. OF SOLUTION. SP. GR. OF DRY SLIME = 2·7.

Diameter of Vat in Feet.	Circumference of Vat in Feet.	Cubic Feet per Foot Depth. Area of Vat in Square Feet.	Tons of Dry Sand per Foot Depth. Collected.	Tons of Dry Sand per Foot Depth. Leached.	Tons of Dry Sand per Foot Depth. Transferred.	Tons of Solution per Foot Depth.	Slime Pulp. Percentage of Moisture. 30%	32·5%	35%	37·5%	40%	42·5%	45%	50%	1 to 1½	1 to 2	1 to 2½	1 to 3	1 to 3½	1 to 4	1 to 4½	1 to 5	Ratio of Dry Slime to Solution.
							1·788	1·739	1·693	1·649	1·607	1·567	1·530	1·459	1·337	1·266	1·219	1·187	1·163	1·144	1·129	1·117	Sp. Gr. of Pulp
			Cubic Feet per Ton.				25·56	27·27	29·08	31·05	33·19	35·35	38·01	43·89	59·79	75·75	92·00	107·75	123·62	139·91	156·18	172·20	Cub. feet of Pulp per ton Dry Slime.
			21·5	24	26	32	Tons of Dry Slime per Foot Depth of Pulp.																Diam. of Vat in feet.
5	15·71	19·63	·913	·818	·755	·613	·768	·720	·675	·632	·591	·552	·516	·447	·328	·259	·213	·182	·159	·140	·126	·114	5
6	18·83	28·27	1·31	1·18	1·09	·883	1·106	1·037	·972	·910	·852	·710	·744	·644	·472	·373	·307	·262	·228	·202	·181	·164	6
7	21·99	38·48	1·79	1·60	1·48	1·20	1·31	1·411	1·32	1·240	1·16	1·083	1·01	·876	·643	·508	·418	·357	·311	·275	·246	·223	7
8	25·13	50·27	2·34	2·09	1·93	1·57	1·96	1·843	1·73	1·620	1·31	1·415	1·32	1·15	·841	·664	·546	·466	·407	·359	·322	·292	8
9	28·27	63·63	2·96	2·65	2·45	1·98	2·49	2·333	2·19	2·049	1·92	1·791	1·67	1·45	1·06	·840	·691	·590	·515	·455	·407	·369	9
10	31·42	78·54	3·65	3·27	3·02	2·45	3·07	2·880	2·70	2·529	2·36	2·211	2·06	1·79	1·31	1·03	·855	·728	·633	·561	·503	·456	10
11	34·56	95·03	4·42	3·96	3·66	2·97	3·72	3·485	3·27	3·061	2·86	2·675	2·50	2·17	1·59	1·25	1·03	·882	·769	·679	·608	·552	11
12	37·70	113·10	5·26	4·71	4·35	3·53	4·42	4·147	3·89	3·643	3·41	3·183	2·98	2·58	1·89	1·49	1·23	1·64	·915	·808	·724	·657	12
13	40·84	132·73	6·17	5·53	5·11	4·15	5·19	4·867	4·56	4·275	4·00	3·736	3·49	3·02	2·22	1·75	1·44	1·23	1·07	·949	·850	·771	13
14	43·98	153·94	7·16	6·41	5·92	4·81	6·02	5·645	5·29	4·958	4·64	4·333	4·05	3·51	2·57	2·03	1·67	1·43	1·25	1·10	·986	·894	14
15	47·12	176·72	8·22	7·36	6·80	5·52	6·91	6·480	6·07	5·691	5·32	4·974	4·65	4·03	2·95	2·33	1·92	1·64	1·43	1·26	1·13	1·03	15
16	50·27	201·06	9·35	8·38	7·73	6·28	7·87	7·373	6·91	6·475	6·06	5·659	5·29	4·58	3·36	2·65	2·19	1·87	1·63	1·44	1·29	1·17	16
17	53·41	226·98	10·55	9·46	8·73	7·09	8·88	8·323	7·81	7·310	6·84	6·388	5·97	5·17	3·80	3·00	2·47	2·11	1·84	1·62	1·45	1·32	17
18	56·55	254·47	11·84	10·60	9·78	7·95	9·96	9·331	8·75	8·195	7·67	7·162	6·69	5·80	4·26	3·36	2·77	2·36	2·06	1·82	1·63	1·48	18
19	59·69	283·53	13·19	11·81	10·91	8·86	11·09	10·397	9·75	9·131	8·54	7·980	7·46	6·46	4·74	3·74	3·08	2·63	2·29	2·03	1·82	1·65	19
20	62·83	314·16	14·61	13·09	12·08	9·82	12·28	11·520	10·80	10·118	9·47	8·842	8·26	7·16	5·25	4·13	3·42	2·91	2·54	2·24	2·01	1·83	20
21	65·97	346·36	16·11	14·43	13·32	10·82	13·55	12·701	11·91	11·155	10·44	9·748	9·11	7·89	5·79	4·57	3·76	3·21	2·80	2·48	2·22	2·01	21
22	69·12	380·13	17·68	15·84	14·62	11·88	14·87	13·939	13·07	12·243	11·43	10·699	10·00	8·66	6·36	5·02	4·13	3·53	3·07	2·72	2·43	2·21	22
23	72·26	415·48	19·32	17·31	15·98	12·98	16·26	15·235	14·29	13·381	12·32	11·694	10·93	9·47	6·95	5·48	4·52	3·86	3·36	2·97	2·66	2·41	23
24	75·40	452·39	21·04	18·85	17·40	14·14	17·70	16·589	15·56	14·570	13·63	12·733	11·90	10·31	7·57	5·97	4·92	4·20	3·66	3·23	2·90	2·63	24
25	78·54	490·87	22·83	20·45	18·88	15·34	19·19	18·000	16·88	15·809	14·79	13·816	12·90	11·19	8·21	6·47	5·35	4·55	3·96	3·51	3·15	2·86	25
26	81·68	530·93	24·69	22·12	20·42	16·59	20·77	19·469	18·26	17·099	16·00	14·943	13·97	12·10	8·88	7·01	5·77	4·93	4·29	3·79	3·40	3·08	26
27	84·82	572·56	26·63	23·86	22·02	17·89	22·40	20·996	19·59	18·440	17·23	16·115	15·06	13·05	9·58	7·56	6·22	5·31	4·63	4·09	3·67	3·32	27
28	87·97	615·75	28·64	25·66	23·68	19·24	24·03	22·580	21·17	19·831	18·55	17·330	16·20	14·03	10·30	8·13	6·69	5·71	4·98	4·40	3·94	3·58	28
29	91·11	660·52	30·72	27·52	25·40	20·64	25·84	24·221	22·71	21·273	19·90	18·590	17·38	15·05	11·05	8·72	7·18	6·13	5·34	4·72	4·23	3·84	29
30	94·25	706·86	32·87	29·45	27·19	22·09	27·64	25·921	24·30	22·765	21·30	19·895	18·62	16·12	11·81	9·31	7·70	6·56	5·70	5·05	4·53	4·11	30
32·5	102·10	829·58	38·59	34·57	31·91	25·92	32·46	30·421	28·53	26·718	24·99	23·340	21·83	18·90	13·87	10·95	9·02	7·70	6·71	5·93	5·31	4·82	32·5
35	109·96	962·11	44·74	40·09	37·00	30·07	37·62	35·281	33·09	30·986	28·99	27·079	25·28	21·94	16·00	12·68	10·48	8·92	7·76	6·88	6·17	5·60	35
37·5	117·80	1104·47	51·37	46·02	42·48	34·31	43·21	40·501	37·98	35·571	33·28	31·086	29·06	25·16	18·47	14·58	12·01	10·23	8·93	7·89	7·07	6·41	37·5
40	125·66	1256·64	58·44	52·36	48·33	39·27	48·13	46·081	43·20	40·471	37·86	35·368	33·02	28·63	21·00	16·56	13·60	11·63	10·14	8·98	8·05	7·31	40
42·5	133·26	1413·06	65·72	58·88	54·35	44·16	55·28	51·817	48·59	45·309	42·57	39·771	37·18	32·20	23·63	18·65	15·36	13·11	11·43	10·10	9·05	8·21	42·5
45	141·37	1590·43	73·97	66·27	61·17	49·70	62·19	58·322	54·69	31·222	48·24	44·703	41·80	36·20	26·58	20·95	17·32	14·75	12·83	11·37	10·19	9·25	45
47·5	149·23	1772·05	82·42	73·84	68·16	55·38	69·33	64·982	60·94	57·071	53·39	49·875	46·62	40·37	29·64	23·39	19·26	16·45	14·33	12·67	11·35	10·29	47·5
50	157·08	1963·50	91·32	81·81	75·52	61·36	76·77	72·002	67·52	63·237	59·16	55·263	51·00	44·77	32·81	25·87	21·39	18·21	13·84	14·04	12·58	11·42	50
52·5	164·93	2164·75	100·69	90·20	83·26	67·65	84·69	79·382	74·44	69·718	65·22	60·927	56·85	49·32	36·21	28·38	23·53	20·09	17·51	15·47	13·86	12·57	52·5
55	172·79	2375·83	110·50	99·00	91·38	74·24	92·90	87·122	81·70	76·516	71·58	66·868	62·44	54·17	39·71	31·30	25·88	22·03	19·17	16·90	15·23	13·82	55
57·5	180·64	2596·72	120·78	108·20	99·87	81·15	101·59	95·223	89·30	83·630	78·24	73·082	68·32	59·16	43·43	34·28	28·23	24·10	21·01	18·56	16·63	15·08	57·5
60	188·50	2827·43	131·50	117·81	108·75	88·36	110·56	103·683	97·20	91·061	85·19	79·579	74·31	64·47	47·25	37·25	30·80	26·22	22·82	20·22	18·12	16·45	60
65	204·20	3318·31	154·34	138·26	127·63	103·70	129·82	121·684	114·11	106·870	99·88	93·395	87·30	75·61	55·50	43·81	36·07	30·80	26·84	23·72	21·25	19·27	65
70	219·91	3848·45	178·99	160·35	148·02	120·26	150·49	141·124	132·36	123·944	115·95	108·316	101·14	87·74	64·31	50·70	41·92	35·60	31·05	27·52	24·66	22·39	70
75	235·6	4417·86	205·48	184·08	169·91	138·06	172·84	162·004	151·92	142·282	133·11	124·342	116·23	100·66	73·89	58·32	48·02	41·00	35·74	31·58	28·29	25·66	75
80	251·33	5026·55	233·79	209·43	193·32	157·07	196·55	184·325	172·80	161·886	151·29	141·473	132·20	114·10	83·94	66·35	54·29	46·24	40·21	35·69	32·17	29·15	80
85	267·04	5674·50	263·93	236·44	218·25	177·33	222·01	205·088	193·13	182·734	170·97	159·710	149·29	129·29	94·91	74·91	61·68	52·66	45·90	40·56	36·33	32·93	85
90	282·74	6361·73	295·89	265·07	244·68	198·80	248·76	233·287	218·79	204·887	191·48	179·052	167·31	144·41	106·23	83·97	68·70	58·33	50·89	45·17	40·71	36·90	90
95	298·45	7088·22	329·68	295·34	272·62	221·51	277·32	259·927	243·75	228·284	213·56	199·300	186·48	161·30	118·55	93·57	77·05	63·78	57·34	50·66	45·38	41·16	95
100	314·16	7853·98	365·30	327·25	302·07	245·43	307·11	298·008	270·08	252·946	236·40	221·052	206·56	178·92	131·16	103·67	84·82	72·26	62·83	55·76	50·27	45·55	100

*From *Jour.* Chem. Met. & Min. Soc., South Africa.

TABLE FOR STANDARDIZING SUMP SOLUTIONS

By C. W. Hess

(October 1, 1910)

	1	2	3	4	5	6	7	8	9	10
01	-3	-6	-8	-13	1	1-3	1-6	1-8	1-13	2
02	-6	-13	1-3	1-8	2	2-6	2-13	3-3	3-8	4
03	-8	1-3	1-13	2-6	3	3-8	4-3	4-13	5-6	6
04	-13	1-8	2-6	3-3	4	4-13	5-8	6-6	7-3	8
.05	1	2	3	4	5	6	7	8	9	10
.06	1-3	2-6	3-8	4-13	6	7-3	8-6	9-8	10-13	12
.07	1-6	2-13	4-3	5-8	7	8-6	9-13	11-3	12-8	14
.08	1-8	3-3	4-13	6-6	8	9-9	11-3	12-13	14-6	16
.09	1-13	3-8	5-6	7-3	9	10-13	12-8	14-6	16-3	18
.10	2	4	6	8	10	12	14	16	18	20
.11	2-3	4-6	6-8	8-13	11	13-3	15-6	17-8	19-13	22
.12	2-6	4-13	7-3	9-8	12	14-6	16-13	19-3	21-8	24
.13	2-8	5-3	7-13	10-6	13	15-8	17-3	20-13	23-6	26
.14	2-13	5-8	8-6	11-3	14	16-13	19-8	22-6	25-3	28
.15	3	6	9	12	15	18	21	24	27	30
.16	3-3	6-6	9-8	12-13	16	19-3	22-6	25-8	28-13	32
.17	3-6	6-13	10-3	13-8	17	20-6	23-13	27-3	30-8	34
.18	3-8	7-3	10-13	14-6	18	21-8	25-3	28-13	32-6	36
.19	3-13	7-8	11-6	15-3	19	22-13	26-8	30-6	34-3	38
.20	4	8	12	16	20	24	28	32	36	40
.21	4-3	8-6	12-8	16-13	21	25-3	29-6	33-8	37-13	42
.22	4-6	8-13	13-3	17-8	22	26-6	30-13	35-3	39-8	44
.23	4-8	9-3	13-13	18-6	23	27-8	32-3	36-13	41-6	46
.24	4-13	9-8	14-6	19-3	24	28-13	33-8	38-6	43-3	48
.25	5	10	15	20	25	30	35	40	45	50
.26	5-3	10-6	15-8	20-13	26	31-3	36-6	41-8	46-13	52
.27	5-6	10-13	16-3	21-8	27	32-6	37-13	43-3	48-8	54
.28	5-8	11-3	16-13	22-6	28	33-8	40-3	44-13	50-6	56
.29	5-13	11-8	17-6	23-3	29	34-13	41-8	46-6	52-3	58
.30	6	12	18	24	30	36	42	48	54	60
.31	6-3	12-6	18-8	24-13	31	37-3	43-6	49-8	55-13	62
.32	6-6	12-13	19-3	25-8	32	38-6	44-13	51-3	57-8	64
.33	6-8	13-3	19-13	26-6	33	39-8	46-3	52-13	59-6	66
.34	6-13	13-8	20-6	27-3	34	40-13	47-8	54-4	61-3	68
.35	7	14	21	28	35	42	49	56	63	70
.36	7-3	14-6	21-8	28-13	36	43-3	50-6	57-8	64-13	72
.37	7-6	14-13	22-3	29-8	37	44-6	51-13	59-3	66-8	74
.38	7-8	15-3	22-13	30-6	38	45-8	53-3	60-13	68-6	76
.39	7-13	15-8	23-6	31-3	39	46-13	54-8	62-6	70-3	78
.40	8	16	24	32	40	48	56	64	72	80
.41	8-3	16-6	24-8	32-13	41	49-3	57-6	65-8	73-13	82
.42	8-6	16-13	25-3	33-8	42	50-6	57-13	67-3	75-8	84
.43	8-8	17-3	25-13	34-6	43	51-8	60-3	68-13	77-6	86
.44	8-13	17-8	26-6	35-3	44	52-13	61-8	70-6	79-3	88
.45	9	18	27	36	45	54	63	72	81	90
.46	9-3	18-6	27-8	36-13	46	55-3	64-6	73-8	82-13	92
.47	9-6	18-13	28-3	37-8	47	56-6	65-13	75-3	84-8	94
.48	9-8	19-3	28-13	38-6	48	57-8	67-3	76-13	86-6	96
.49	9-13	19-8	29-6	39-3	49	58-13	68-8	78-6	88-3	98
.5	10	20	30	40	50	60	70	80	90	100

	11	12	13	14	15	16	17	18	19	20
01	2-3	2-6	2-8	2-13	3	3-3	3-6	3-8	3-13	4
02	4-6	4-13	5-3	5-8	6	6-6	6-13	7-3	7-8	8
03	6-8	7-3	7-13	8-6	9	9-8	10-3	10-13	11-6	12
04	8-13	9-8	10-6	11-3	12	12-8	13-6	14-3	14-13	16
.05	11	12	13	14	15	16	17	18	19	20
.06	13-3	14-6	15-8	16-13	18	19-3	20-6	21-8	22-13	24
.07	15-6	16-13	18-3	19-9	21	22-6	23-13	25-3	26-8	28
.08	17-8	19-3	20-13	22-6	24	25-8	27-3	28-13	30-6	32
.09	19-13	21-8	23-6	25-3	27	28-13	30-8	32-6	34-3	36
.10	22	24	26	28	30	32	34	36	38	40
.11	24-3	26-6	28-8	30-13	33	35-3	37-6	39-8	41-13	44
.12	24-6	28-13	31-3	33-8	36	38-6	41-13	43-3	45-8	48
.13	28-8	31-3	33-13	36-6	39	41-8	45-3	46-13	49-6	52
.14	30-13	33-8	36-6	39-3	42	44-13	48-8	50-6	53-3	56
.15	33	36	39	42	45	48	51	54	57	60
.16	35-3	38-6	41-8	44-13	48	51-3	54-6	57-8	60-13	64
.17	37-6	40-13	44-3	47-8	51	54-6	57-13	61-3	64-8	68
.18	39-8	43-3	46-13	50-6	54	57-8	61-3	64-13	66-6	72
.19	41-13	45-8	49-6	53-3	57	60-13	64-8	68-6	72-3	76
.20	44	48	52	56	60	64	68	72	76	80
.21	46-3	50-6	54-8	58-13	63	67-3	71-6	75-8	79-13	84
.22	48-6	52-13	57-3	61-8	66	70-6	74-13	79-3	83-8	88
.23	50-8	55-3	59-13	64-6	69	73-8	78-3	82-13	87-6	92
.24	52-13	57-6	62-6	67-3	72	74-13	81-8	86-6	91-3	96
.25	55	60	65	70	75	80	85	90	95	100
.26	57-3	62-6	67-8	73-13	78	83-3	88-6	93-8	98-13	104
.27	59-6	64-13	70-3	75-8	81	86-6	91-13	97-3	102-8	108
.28	61-8	67-3	72-13	78-6	84	89-8	95-3	100-13	106-6	112
.29	63-13	69-8	75-6	81-3	87	92-13	98-8	104-6	110-3	116
.30	66	72	78	84	90	96	102	108	114	120
.31	68-3	74-6	80-8	86-13	93	99-3	105-6	111-8	117-13	124
.32	70-6	76-13	83-3	89-8	96	102-6	108-13	115-3	121-8	128
.33	72-8	81-3	85-13	92-6	99	105-8	112-3	118-13	125-6	132
.34	74-13	83-8	88-6	95-3	102	108-13	115-8	122-6	129-3	136
.35	77	84	91	98	105	112	119	126	133	140
.36	79-3	86-6	93-8	100-13	108	115-3	122-6	129-8	136-13	144
.37	81-6	88-13	96-3	103-8	111	118-6	125-13	133-3	140-8	148
.38	83-8	91-3	98-13	106-6	114	121-8	129-3	136-13	144-6	152
.39	85-13	93-8	101-6	109-3	117	124-13	132-8	140-6	148-3	156
.40	88	96	104	112	120	128	136	144	152	160
.41	90-3	98-6	106-8	114-13	123	131-3	139-6	147-8	155-13	164
.42	92-6	101-13	109-3	117-8	126	134-6	142-13	151-3	159-8	168
.43	94-8	103-3	111-13	120-6	129	137-8	146-3	154-13	163-6	172
.44	96-13	105-8	114-6	123-3	132	140-13	149-8	158-6	167-3	176
.45	99	108	117	126	135	144	153	162	171	180
.46	101-3	110-6	119-8	128-13	138	147-3	156-6	165-8	174-13	184
.47	103-6	112-13	122-3	131-8	141	150-6	159-13	169-3	178-8	188
.48	105-8	115-3	124-13	134-6	144	153-8	163-3	172-13	182-6	192
.49	107-13	117-8	127-6	137-3	147	158-13	166-8	176-6	186-3	196
.5	110	120	130	140	150	160	170	180	190	200

	21	22	23	24	25	26	27	28	29	30
01	4-3	4-6	4-8	4-13	5	5-3	5-6	5-8	5-13	6
02	8-6	8-13	9-3	9-6	10	10-6	10-13	11-3	11-8	12
03	12-8	13-3	13-13	14-6	15	15-8	16-3	16-13	17-6	18
04	16-13	17-8	18-6	19-3	20	20-13	21-8	22-6	23-3	24
.05	21	22	23	24	25	26	27	28	29	30
.06	25-3	26-6	27-8	28-13	30	31-3	32-6	33-8	34-13	36
.07	29-6	30-13	32-3	33-8	35	36-6	37-13	39-3	40-8	42
.08	33-8	35-3	36-13	38-6	40	41-8	43-3	44-13	46-6	48
.09	37-13	39-8	41-6	43-3	45	46-13	48-8	50-6	52-3	54
.10	42	44	46	48	50	52	54	55	58	60
.11	46-3	48-6	50-8	52-13	55	57-3	59-6	61-8	63-13	66
.12	50-6	52-13	55-3	57-8	60	62-6	64-13	67-3	69-8	72
.13	54-8	57-3	59-13	62-6	65	67-8	70-3	72-13	75-6	78
.14	58-13	61-8	64-6	67-3	70	72-13	75-8	78-6	81-3	84
.15	63	66	69	72	75	78	81	84	87	90
.16	67-3	70-6	73-8	76-13	80	83-3	86-6	89-8	92-13	96
.17	71-6	74-13	78-3	81-8	85	88-6	91-13	95-3	98-8	102
.18	75-8	79-3	82-13	86-6	90	93-8	97-3	100-13	104-6	108
.19	79-13	83-8	87-6	91-3	95	98-13	101-8	106-6	110-3	114
.20	84	88	92	96	100	104	108	112	116	120
.21	88-3	92-6	96-8	100-13	105	109-3	113-6	117-8	121-13	126
.22	92-6	96-13	101-3	105-8	110	114-6	118-13	123-3	127-8	132
.23	96-8	101-3	105-13	110-6	115	119-8	124-3	128-13	133-6	138
.24	100-13	105-8	110-6	115-3	120	124-13	129-8	134-6	139-3	144
.25	105	110	115	120	125	130	135	140	145	150
.26	109-3	114-6	119-8	124-13	130	135-3	140-6	145-8	150-13	156
.27	113-6	118-13	124-3	129-8	135	140-6	145-13	151-3	156-8	162
.28	117-8	123-3	128-13	134-6	140	145-8	151-3	156-13	162-6	168
.29	121-13	127-8	123-6	139-3	145	150-13	156-8	162-6	168-3	174
.30	126	132	138	144	150	156	162	168	174	180
.31	130-3	136-6	142-8	148-13	155	161-3	167-6	173-8	179-13	186
.32	134-6	140-13	147-3	153-8	160	166-6	172-13	179-3	185-8	192
.33	138-8	145-3	151-13	158-6	165	171-8	178-3	184-13	191-6	198
.34	142-13	149-8	156-6	163-3	170	176-13	183-8	190-6	197-3	204
.35	147	154	161	168	175	182	189	196	203	210
.36	151-3	158-3	165-8	172-13	180	187-3	194-6	201-8	208-13	216
.37	155-6	162-13	170-3	177-8	185	192-6	199-13	207-3	214-8	222
.38	159-8	167-3	174-13	182-6	190	197-8	205-3	212-13	220-6	228
.39	163-13	171-8	179-6	187-3	195	202-13	210-7	218-6	226-3	234
.40	168	176	184	192	200	208	216	224	232	240
.41	172-3	180-6	188-8	196-13	205	213-3	221-6	229-8	237-13	246
.42	176-6	184-13	193-3	201-8	210	218-6	226-13	235-2	243-8	252
.43	180-8	189-3	197-13	206-6	215	223-8	232-3	240-13	249-6	258
.44	184-13	193-8	202-6	211-3	220	228-13	237-8	246-6	255-2	264
.45	189	198	207	216	225	234	243	252	261	270
.46	193-3	202-6	211-8	220-13	230	239-3	248-6	257-8	266-13	276
.47	197-6	206-13	216-3	225-8	235	246-6	253-13	263-3	272-8	282
.48	210-8	211-3	202-13	230-6	240	249-8	259-3	268-13	278-6	288
.49	214-13	215-8	207-6	236-3	248	254-13	264-8	274-6	284-3	294
.5	210	220	230	240	250	260	270	280	290	300

The table on the opposite page shows the quantity of KCN necessary in any given number of tons of solution from one to thirty at strengths varying from 0.01 to 0.5%. This table has proved convenient for standardizing sump solutions. The top row of figures indicates the tons of solution to be made up to a standard strength and the decimals in the left-hand column are the strengths of solution. The number of pounds and ounces necessary to be added to the solution will always be found in the angle of the lines of these figures. For example, to find the number of pounds of cyanide necessary to add to 13 tons of solution to raise its strength 0.17%, follow down the column headed by 13 to the line on which the decimal 0.17 is found. The figures 44-3 there found, give the answer, 44 pounds and 3 ounces.

It is convenient to have indicators on the solution tanks marked in tons in preference to feet, since solutions are recorded in tons when calculations are made for standardizing or for running a given quantity on a charge of ore or for making pulp.

PRACTICAL APPLICATIONS OF THE SPECIFIC GRAVITY FLASK

By H. STADLER

(January 27, 1912)

*The principal objects of these notes is to show the great practical possibilities of the specific gravity flask, which in future should prove to be one of the most useful and indispensable instruments for efficient and reliable control of the condition of mill-pulp. W. J. Sharwood's admirable article on the measurement of pulp and tailing[1] is presumably familiar to all millmen who have to deal practically with the handling of pulp. The present notes do not pretend to add much that is really novel, but I trust that some practical conclusions drawn from this and other papers on this subject[2] will be welcome to those whose daily duties leave very little spare time for a thorough study of technical literature.

In practice it has been found that flasks having a large mouth ground true, the ground edge to be used as the mark of capacity, are more convenient for taking fair samples than the usual glasses with a mark below the mouth. In consequence of sand adhering to the glass above the mark, the latter are more difficult to fill accurately.

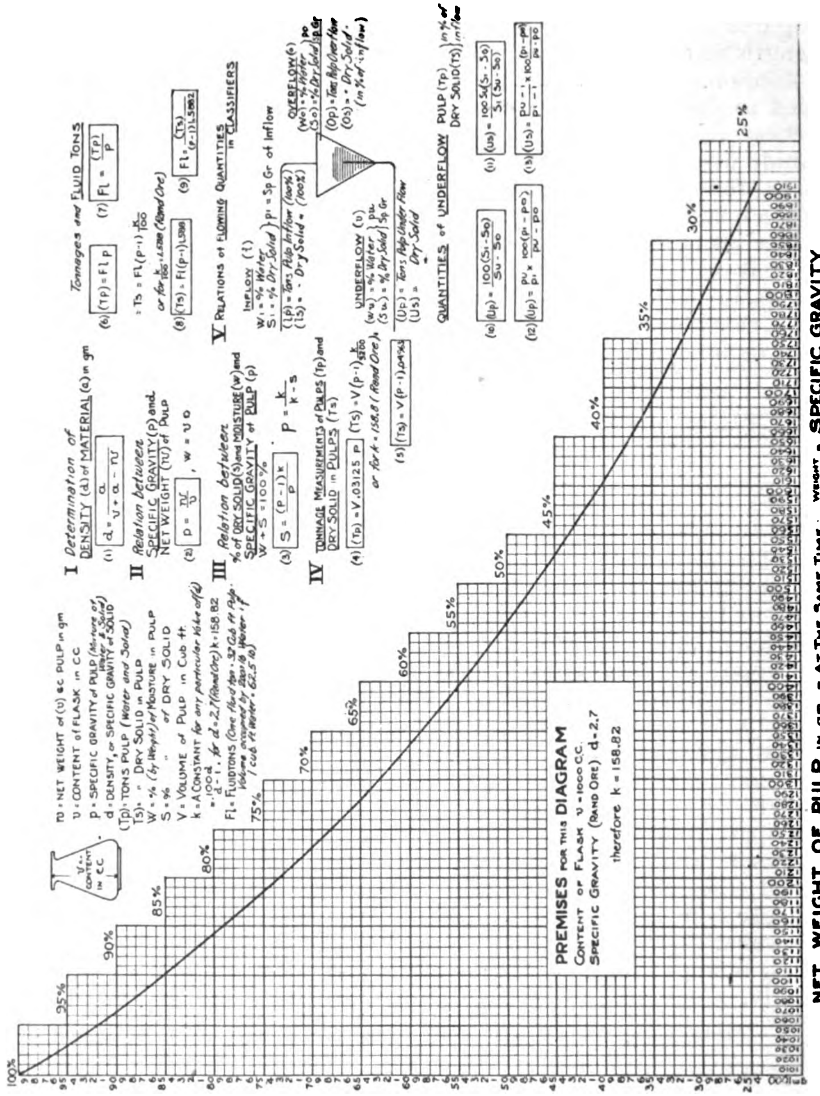
I. Determination of Density (Specific Gravity) of Solids

If the specific gravity of the solid ore under treatment be not taken at an accepted average value, it may be determined by any

*Extract from the *Jour.* Chem., Met. and Min. Soc. of South Africa, November, 1911.

[1]*The Mining Magazine,* Vol. I, Nov. and Dec. 1909.

[2]See F. B. Hyder's 'Specific Gravity Estimation of Pulp,' Colorado Scientific Society, Vol. IX, p. 417; and W. A. Caldecott, *Jour.* Chem., Met. and Min. Soc. of S. A., Vol II, p. 375.


PERCENTAGES OF MOISTURE
NET WEIGHT OF PULP IN gm - AT THE SAME TIME WEIGHT/1000 = SPECIFIC GRAVITY
100% 95% 90% 85% 80% 75% 70% 65% 60% 55% 50% 45% 40% 35% 30% 25%
V = CONTENT IN C.C.
w = NET WEIGHT of (v) cc PULP in gm
v = CONTENT of FLASK in CC
p = SPECIFIC GRAVITY of PULP (Mixture of Water & Solid)
d = DENSITY, or SPECIFIC GRAVITY of SOLID
(Tp) = TONS PULP (Water and Solid)
(Ts) = " DRY SOLID in PULP
W = % (by Weight) of MOISTURE in PULP
S = % " of DRY SOLID
V = VOLUME of PULP in Cub ft.
k = A CONSTANT for any particular Value of (d) = 100d/(d - 1), for d = 2.7 (Rand Ore) k = 158.82
Fl = FLUIDTONS (One fluid ton = 32 Cub ft Pulp. Volume occupied by 2000 lb Water; 1 cub ft Water = 62.5 lb)
I Determination of DENSITY (d) of MATERIAL (a) in gm
(1) d = a/(v + a - w)
II Relation between SPECIFIC GRAVITY (p) and NET WEIGHT (w) of PULP
(2) p = w/v, w = v p
III Relation between % of DRY SOLID (S) and MOISTURE (W) and SPECIFIC GRAVITY of PULP (p)
W + S = 100%
(3) S = (p - 1)k/p   p = k/(k - S)
IV TONNAGE MEASUREMENTS of PULPS (Tp) and DRY SOLID in PULPS (Ts)
(4) (Tp) = V .03125 p   (Ts) = V (p - 1) k/3200
or for k = 158.8 (Rand Ore),
(5) (Ts) = V (p - 1) .04963
Tonnages and FLUID TONS
(6) (Tp) = Fl p   (7) Fl = (Tp)/p
∴ Ts = Fl (p - 1) k/100
or for k/100 = 1.588 (Rand Ore)
(8) (Ts) = Fl (p - 1) 1.588   (9) Fl = (Ts)/((p - 1) 1.5882)
V RELATIONS of FLOWING QUANTITIES in CLASSIFIERS
INFLOW (i)
Wi = % Water
Si = % Dry Solid } pi = Sp Gr of Inflow
(Ip) = Tons Pulp Inflow (100%)
(Is) = " Dry Solid " (100%)
OVERFLOW (o)
(Wo) = % Water
(So) = % Dry Solid } po Sp Gr
(Op) = Tons Pulp Overflow
(Os) = " Dry Solid " (in % of inflow)
UNDERFLOW (u)
(Wu) = % Water
(Su) = % Dry Solid } pu Sp Gr
(Up) = Tons Pulp Under Flow
(Us) = " Dry Solid
QUANTITIES of UNDERFLOW PULP (Tp) / DRY SOLID (Ts) } in % of inflow
(10) (Up) = 100(Si - So)/(Su - So)   (11) (Us) = 100 Su(Si - So)/(Si(Su - So))
(12) (Up) = pu/pi × 100(pi - po)/(pu - po)   (13) (Us) = (pu - 1)/(pi - 1) × 100(pi - po)/(pu - po)
PREMISES FOR THIS DIAGRAM
CONTENT OF FLASK v = 1000 C.C.
SPECIFIC GRAVITY (RAND ORE) d = 2.7
therefore k = 158.82

well known method. For instance, if the material is dry, a known weight (in grams) of the material under examination is placed in the flask, and water added to the mark (in the suggested flask to the top), the total weight of the contents is then taken. The density is obtained by the use of formula (1). Wet material, such as mill-pulp, is preferably dried and weighed after the determination of the specific gravity of the pulp.

II. Determination of Specific Gravity of Pulps

Almost all factors determining the nature of pulps, from the millman's point of view, are governed by the specific gravity of the pulp. If the metric system of weights be used, the specific gravity is at once obtained by dividing the net weight of the pulp by the volume occupied, or when a 1000-c.c. flask is used by cutting off the last three decimals of the net weight of the pulp (in grams). The specific gravity of slime pulp, in which all solids are freely suspended, may be measured by a hydrometer.

III. Determination of Precentage (by Weight) of Dry Solid and Water

Although the specific gravity is indicative of the constitution of pulps, the percentage of dry solid or moisture conveys a more definite impression to the average mind. The usual method of determining the percentage of solids and moisture, by drying and weighing, is very troublesome, and also, as done in practice, unreliable and inaccurate. Hence methods based on specific gravity, formula (3), are vastly quicker and more convenient.

Any alteration in the composition and in the rate of flow of pulps shows itself by a marked change in the percentage of moisture and this therefore gives a reliable, sensitive, and effective control in the uniform distribution of the pulp and the regular working of classifiers. For instance, at a mine where a set of eight similar classifiers was fed with the same battery pulp, the moisture in the underflow of the different classifiers varied, at the same moment, from 35 to 75%, thus showing that the distribution of the pulp over the classifiers was very irregular. It is to be hoped that, with such a convenient means at hand, millmen will pay more attention to the control of individual units, as the total average always gives figures which are no help in detecting faulty work.

IV. Tonnage Measurements of Total Pulps (Tp) and of Dry Solid in Pulps (Ts)

Tonnage measurements of flowing quantities are generally made by running pulp for a measured time into a vessel of sufficient capacity to give fairly accurate results. Scales for weighing the large quantities collected are not practically available, and the drawing off of the water, drying and weighing of the solid portion

in such quantities still less practicable. All these inconveniences are avoided by reckoning the tonnage from the volume occupied by the total pulp (in cubic feet) and its specific gravity, by formula (4) for the total pulp, and by formula (5) for the dry solid portion only.

The simplest manner to carry out these measurements is to run the pulp into a box of known capacity, large enough to take at least one minute's run of the stream. The exact time required for filling is taken by a stop-watch, the point of overflow being easily and exactly determinable. The specific gravity is determined from separate representative samples taken simultaneously, and the calculated weight of pulp or dry slime passing during the time (t in seconds) are then used to give the hourly or daily quantities. If, in all measurements, boxes of equal capacity are used, for instance, 16 cu. ft. (the volume of half a ton of water), all the constant factors may be condensed into one. For a box of the above capacity the formulæ for Rand ore are:

$$\text{Tons of total pulp in 24 hours: (Tp)} = \frac{43{,}200p}{t}$$

$$\text{Tons of dry solid only (Ts)} = \frac{(p-1)\ 68609}{t}$$

Since the capacities of launders, vats, pumps, and other machines, as well as the rates of overflow of classifiers are determined by the volume of the pulp, the fluid ton of 32 cu. ft. (equal to the volume of 2000 lb. water, on the assumption that 1 cu. ft. of water weighs 62.5 lb.) is frequently used as a unit in mining work. The tonnage of pulps of known specific gravities is readily converted into fluid tons by formula (7) for the total pulp and by formula (9) for the dry solid portion.

The same method is also applicable on a large scale to the tonnage measurement of vats, provided the consistence of the pulp is not changed by drawing off water or slime (as is the case with sand-collecting and slime-settling vat), but by adding cyanide solution (as happens with slime-treatment vats). For large measurements the practical utility of this method is therefore confined to agitation or circulating vats, and in all other cases, for lack of better methods, resort must be had to the method commonly practiced, inaccurate as it is, of estimating the dry solid by the established dry weight of the settlement volume. This varies considerably with the nature and the fineness of the ore, and should therefore be checked frequently and corrected if necessary by actual test. For such tests the method I described is suitable, as no drawing off of water nor drying is necessary.

At the Knight Central G. M. Co. the daily tonnage measurements of the mill-pulp for several years past have been regularly taken by the gravity method, which, after careful investigation, was found to give most reliable results. In addition there is the advantage that it can be carried out without waiting for settlement of the slime. After filling the first transfer tank with solution and

after circulation (the whole operation taking about four hours), two samples are taken, one at the delivery pipe, representing the pulp at the bottom of the tank, and one from the upper part, to get a representative average for the whole content.

V. Relations of Flowing Quantities in Classifiers

The relation of flowing quantities in classifiers may be calculated from the percentage of any of their components measured at the inflow, overflow, and underflow. No component can so quickly and reliably be measured as the percentage of dry solid taken with the specific gravity flask. From the percentages of dry solid so obtained, the tonnage of the total pulp underflow (as a percentage of the total pulp inflow) is calculated by formula (10) and the tonnage of dry solid underflow (as a percentage of dry solid inflow) by formula (11). In neither case is it first necessary to calculate the percentage of solid, since the flowing quantities of the underflow can directly be determined from the values of the specific gravity, by formula (12) to get the percentage of the total pulp, and by (13) to get that of the dry solid portion only. The accuracy obtainable by the specific gravity method is only limited in practice by the accuracy with which the means at our disposal determine the correct values of (1) the volumetric weight of the pulp (w), (2) the density of the solid (d), and (3) the specific gravity of the liquid (δ).

(1). **Accuracy of Volumetric Weight of Pulp.**—When using a 1000 c.c. flask an error in content or in weighing of even 10 c.c. or 10 gm. respectively, affects the specific gravity (p) of the pulp in the third decimal only. For a 1:1 pulp the error is 0.0007%, which increases with the dilution of the pulp. A variation in (p) of 0.01 in a 1:1 pulp (Rand ore) gives an absolute error of 0.74% in the calculation of the dry solid or moisture percentages, corresponding to 1.48% of the true value. With the dilution of the pulp the error increases progressively. The accuracy of the tonnage measurement of the total pulp varies as that of (p), while the tonnage of the dry portion shows an error of 2.17% which increases progressively with the dilution of the pulp. If we allow such rough work in actual mining practice, the annexed curve gives us the percentage of moisture or solid within ½ to 1 per cent.

(2). **Accuracy of Density of Solid.**—It seems that the average density of Rand ore is, at least for individual mines, fairly steady. The constant (k) in the formula varies with the density as follows:

Density	2.4	2.5	2.6	2.7	2.8	2.9	3.0
k	171.4	166.6	162.5	158.8	155.5	152.6	150.0

A variation from the generally accepted average density of 2.7 for Rand ore to 2.8 affects the accuracy of the specific gravity (p) in a 1:1 pulp by 0.96%, which error decreases slowly with the dilution of the pulp. The tonnage of the total pulp is affected to the same extent as (p), while the dry solid portion shows a constant error

of 2.8% independently of the consistence of the pulp. For the percentage of dry solid and moisture the error is 2.06%, which increases with the dilution of the pulp. When much concentration of the pyritic portion of the ore takes place the actual specific gravity must be ascertained in each case, as an increase from 5 to 10% pyrite causes the specific gravity of the solid to vary from 2.7 to about 2.8 per cent.

(3). **Accuracy of Specific Gravity of Liquid.**—All the formulæ are based on the specific gravity of pure water, but they are applicable to ordinary mixtures of cyanide solutions with enough accuracy for practical purposes, except in cases of excessively thin pulps, such as turbid water, overflow of classifying vats, etc. For exact scientific work or for much heavier liquids (sea water=1.027) the true specific gravity (?), corrected for temperatures, is taken into consideration in the following formulæ:

$$\text{Specific gravity of pulp} = \frac{100\ \delta p}{100d - S\,(d - \delta)}, \text{ and}$$

$$\text{Percentage of dry solid } S = \frac{100\ d\,(p - \delta)}{p\,(d - \delta}.$$

In stating that the extent of the error introduced by ignoring the higher specific gravity of cyanide solutions, is not generally appreciated, Mr. Hyder was misled by erroneous information (accepted by him with the reservation "assuming this to be correct") as to the average density of ordinary cyanide solutions. This strength does not generally exceed 0.18% for sand, 0.025% for slime, and 0.3% for black sand. I am indebted to M. T. Murray, lecturer of the South African School of Mines, for the exact determinations of the specific gravity of an ordinary cyanide solution, assaying 0.126% KCN and 0.01% alkalinity. He obtained a value of 1.0021, and from the data given above it will be seen that the small variation from the specific gravity of pure water is negligible. These results are published through the courtesy of the Mines Trials' Committee, though on my own responsibility.

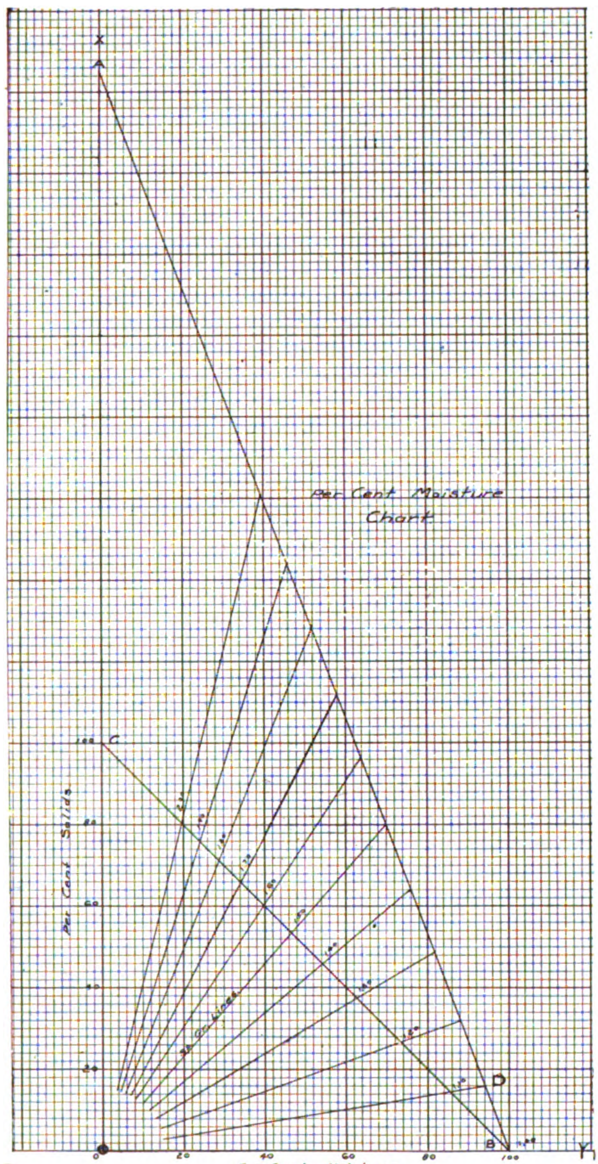
SPECIFIC GRAVITY CHART

(August 3, 1912)

The Editor:

Sir—Frequently in cyanidation, when slime is being handled, the percentage of moisture and the percentage of solids in pulp must be known in order to determine the proper treatment. The common practice is to determine their quantities from the specific gravity determinations, which are taken at frequent intervals during the day. Most moisture percentage charts involve curves and are to a certain extent puzzling. They are constructed by using an algebraic equation and are not easily made. The chart here illustrated is easily made and is clear and correct. It can be made in a few minutes, and with much more detail than shown in the sketch.

The chart is made by using coördinates. The ore in this case has a specific gravity of 2.65. On OY lay off OB, which represents a specific gravity of 1 and which will be the specific gravity when the agitator, or whatever the receptacle containing the pulp, is full of water. Using the same scale, lay off OA on OX, representing



SPECIFIC GRAVITY CHART

the specific gravity of the receptacle when full of solids or ore. This quantity is 2.65. Now the line AB represents all specific gravities ranging from 1 to 2.65. Any intermediate specific gravity lines such as those shown on the chart may be found by dividing this line AB into 16.5 equal parts. (The difference between 2.65 and 1 is 1.65, and the lines on the chart represent one-tenth of 1%, therefore divide by 16.5 instead of 1.65 to get the division, such as BD on the chart.) An equation containing two variables whose sum equals a constant is represented by a straight line 45° to the ordinate. This is the case of the two variables, percentage of moisture and percentage of solid, the sum of which is always 100%. Let this line be any such line as BC, and let the specific gravity lines cut BC. To read this chart, find the point where the line representing the specific gravity already determined cuts the line BC. Reading directly across from this point to the side scale will give the percentage of solids; directly down will give the percentage of moisture.

This chart can be changed to one for a pulp having an ore of different specific gravity than 2.65. In that case let OA on OX represent this new specific gravity in the same scale with OB.

GEORGE B. McLAIN.

Tom Reed Mine, Oatman, Arizona, July 1.

ESTIMATION OF TONNAGE

By A. W. Allen

(February 24, 1912)

The daily estimation of battery tonnage is a necessity, especially where the manager requires details of approximate yield per ton for the purpose of regulating the output. The choice of a suitable method involves the consideration of existing systems, of which the following will be discussed:

1. By the number of cars trammed to the crushers, or crusher-bins, and an estimated weight of ore content.
2. By the actual weight of the cars and ore trammed to the crushers, or crusher-bins, and a deduction for the weight of the cars.
3. By the number of cars trammed to the battery-bin, after crushing, and an estimated weight of ore content.
4. By the use of an automatically weighing belt-conveyor, discharging into the battery-bin.
5. From the 'running time' and an estimated stamp duty.
6. From the yield in the battery and the difference in assay value of the ore, before and after amalgamation or concentration.

In the first instance, where the tonnage is estimated from the number of cars trammed to the crusher-bin, and an estimated weight, the errors arising may be considerable. The accuracy of the system is influenced by the fact that the cars are seldom uniformly filled, and that the condition of the contents, as to size and composition, is liable to vary within wide limits. It is almost im-

possible to estimate the moisture content with any degree of accuracy, or to obtain a truthful average. An advantage is gained by the direct weighing of the cars and the automatic recording of results. In the latter case the moisture is the only consideration affecting the accuracy of the result.

After the ore has passed the grizzlies and crushers it is in better condition for the purpose of tonnage calculation; but an estimation based on the number of cars trammed is again influenced by the way the cars are filled and the degree of moisture in the contents. The latter determination is now an easier matter, but the result is liable to be affected by systematic under or over-filling of the cars, and any carelessness in recording the number. These latter defects can be remedied by the substitution of an automatic weighing, recording, and distributing belt-conveyor in place of the cars; and the degree of accuracy in the result is then only affected by the accuracy of the machine, and the correctness of the moisture deduction.

The determination of moisture content is an important matter, but receives scant attention in the battery. Even after crushing and automatic weighing, an estimation is no easy matter, since *average* moisture content implies the drying of an *average* sample; and the time taken to obtain such a sample would influence the result, unless impracticable precautions were taken to prevent moisture loss. In the few cases where ore milled is quite uniform, a 'grab' sample might contain an average moisture percentage. Contrary conditions, however, predominate; and sometimes it is necessary to consider the effect of tropical rain on alternate cars of clean quartz and old stope fillings, on their way to the mill. In the latter case even an automatic record of actual weight would be exceedingly difficult to correct for moisture content.

All the methods discussed give approximations of tonnage delivered into the crusher or battery-bin, and not tonnage milled. To make daily correction, involving the estimation of what remains in the battery-bin, or both battery-bin and crusher-bin, reduces the estimate, in the majority of cases, to mere guesswork. A monthly estimate would, doubtless, be more correct, since the error liable to arise on the question of storage, would be minimized; but a daily estimation is under consideration. Tonnage totals cannot be added to or subtracted from as the need for correction becomes apparent. At the end of the month the figures must be either taken or rejected as a basis for the compilation of cost, extraction, and other returns.

The daily estimation of tonnage by means of a calculation based upon the 'running time' and estimated stamp duty is at best a loose method, because 'stamp duty' is known to vary considerably and is affected by a number of circumstances which cannot be taken into account in the calculation. In a 200-ton, 50-stamp mill, recovering 30 oz. of silver per ton, and assuming accurate sampling and assaying, a difference of but one-fifth of a ton per stamp per 24 hours between actual and estimated duty would, at the end of a month, mean a difference of no less than 9000 oz. between actual

and calculated returns. The busy millman may also be excused if 'running time' is not always reported in exact figures. This method of calculation has, however, much to recommend it, because no estimation of ore in hand, or moisture content, is needed; but, in the same manner as the results by other methods are affected by variable moisture content, due to lack of uniformity in the product, so is stamp duty liable to fluctuate from the same cause. On the other hand, there are a number of cases where stamp duty remains remarkably uniform, and where this method of calculation is indicated as the best available.

Method No. 6 is, I believe, still in use; and details may serve to prevent further adoption. By this method tonnage is calculated by means of the actual yield in the battery, and the assay of the ore before and after amalgamation or concentration. The figures are bound to tally because the artificial manipulation of results adjusts all possible variations between assay difference and actual returns. The method is only applicable where yield and value are obtainable daily; as, for example, where the concentrate produced is dried, weighed, sampled, and assayed every 24 hours. Given correct sampling and assaying, and the absence of theft or loss in the yield, the result is accurate. On the other hand, with a milled ore yielding high-grade slime and low-grade sand, the extraction figure can be raised at will by the exclusion of a proportion of slime from the tail sample; a drop in stamp duty taking the place of a discrepancy in realization. Theft or mechanical loss would, if undetected, also result in a lower tonnage estimation. In any case, the extraction figures would always agree, and for this reason and because the method is open to abuse, the system is not to be recommended.

From the above considerations it will be seen that, in the absence of consistent uniformity, the correct statement of battery tonnage is a result more of accident than design; and ordinary estimations, although useful for the determination of daily yield per ton in the battery, are valueless for the purpose required in a detailed return showing actual and theoretical extraction. I therefore suggest the daily use of battery estimations for the purpose outlined, but the compilation of extraction and other returns based on figures obtained where the tonnage of individual lots of ore can be checked and re-checked should occasion arise. This is only possible in the cyanide plant where ore units are, generally, of easily defined volume, of uniform composition, and of uniform moisture content.

In leaching-vats, filled by means of automatic distributors and the adjustment of slats, the tonnage is best calculated by the aid of a box with perforated sides and rope handles, of exactly one cubic foot capacity. The perforations in the sides are important and allow the escape of slime as the box fills with sand, uniformly with the vat. As soon as covered the box is pulled up, the contents leveled off and allowed to drain. The box and contents are then weighed, by means of a steel-yard or spring balance. The sand is then tipped out and a moisture sample sent away for immediate estimation. A deduction for the weight of the box is made and the

tonnage in the vat finally calculated from the dry weight of the cubic foot of sand, and the cubic contents of the vat. The result can be verified by alternate weighings.

The tonnage in an agitation-vat is generaaly deduced by methods involving estimations of the specific gravity of both ore and pulp. Published formulae, complicated by the addition of constants and factors dear to the mathematician but abhorrent to the operator, are generally without explanations; and entire reliance on such formulae too often results in mental lassitude and lack of initiative. The correct estimation of ore tonnage in pulp is a much easier matter than is generally supposed; and, if a simple line of reasoning be adopted, the matter of a lost or forgotten formula is of no consequence. The following details may be of use to the working operator who wishes to take an intelligent interest in the method.

The specific gravity of a substance is an amount equal to its weight divided by its volume. Conversely, the volume of a substance is an amount equal to its weight divided by its specific gravity; or is the volume of water it displaces when submerged. Advantage is taken of the latter fact in the preliminary estimation of the specific gravity of the slimed ore, the following method usually being adopted. A suitable flask is taken to which is transferred a portion of slimed ore pulp. The flask is then filled to a mark with water, and weighed. The contents are next discharged, any residue remaining washed out, and the whole carefully dried and weighed. Then if

A be the weight of the dry slimed ore,
B be the weight of flask, slimed ore, and water to mark, and
C be the weight of flask and water to mark,

$$\text{the sp. gr. of the ore} = \frac{\text{Wt. of slimed ore}}{\text{Wt. of water it displaces}} = \frac{A}{(C + A) - B}$$

The result should be carefully checked and the fact remembered that an overestimation of the specific gravity of the ore results, by subsequent calculation, is an underestimation of tonnage; and vice versa.

The specific gravity of the ore pulp in agitation is found in the following way. A fairly wide-mouthed flask or bottle, of known capacity, is taken, filled with the pulp, and weighed. The net weight of the pulp, divided by the volume occupied, will give the specific gravity of the pulp. If a special weight is used to counterbalance the weight of the empty flask, and the capacity of the latter is one litre, the specific gravity of the pulp can be found by direct weighing, and with no more calculation than the placing of a decimal point, thus:

Capacity of flask	1000 c.c.
Wt. of 1 litre of pulp	1330 gm.
Sp. gr. of pulp	1.33

The following line of reasoning may be adopted for the calculation of the percentage of slime, in terms of dry ore, in the pulp:

Vol. of pulp = vol. of ore + vol. of water.

If a = wt. of pulp,
g_1 = sp. gr. of pulp,
x = wt. of ore,
g_2 = sp. gr. of ore,
y = wt. of water,
g_3 = sp. gr. of water,

then

$$\frac{a}{g_1} = \frac{x}{g_2} + \frac{y}{g_3}$$

$$\frac{100}{g_1} = \frac{x}{g_2} + \frac{100-x}{1}$$

$$= \frac{x + 100g_2 - xg_2}{g_2}$$

$$100g_2 = xg_1 + 100g_1g_2 - xg_1g_2$$

$$xg_1g_2 - xg_1 = 100g_1g_2 - 100g_2$$

$$x(g_1g_2 - g_1) = 100g_2(g_1 - 1)$$

$$x = \frac{100g_2(g_1 - 1)}{g_1(g_2 - 1)} = \% \text{ ore in pulp}$$

Given the weight of a cubic foot of water as 62.5 lb., the specific gravity of the pulp, and the capacity of the vat in cubic feet, the estimation of ore content can readily be found by the aid of the calculated percentage of solids. Unnecessary calculations are avoided by filling the agitation vats to the same level, and by the compilation of a table based on the capacity of the vat to this level; and arranged to show, at a glance, the tonnage contained for any average specific gravity of pulp.

Tonnage estimations should be made with the greatest care where the amounts are to be used in conjunction with assay returns. The net weight of ore taken for specific gravity purposes should not be much less than 500 gm.; and weighings should be made on a balance sensitive to 0.5 gm. or under. In specific gravity estimations of the ore, and moisture determinations generally, repeated dryings should be made to demonstrate the absence of residual moisture, the sample being cooled in a desiccator before being weighed.

(February 24, 1912)

The Editor:

Sir—Mr. Stadler's article in your issue of January 27 will have been welcomed by mill and cyanide men who have to obtain tonnage figures from specific gravity measurements of their pulp. If the author could give some hints on how to obtain the value of 'd' (density or solid) in the case of clayey, graphitic, or otherwise non-homogeneous ores, his article would be of yet greater value. A short time ago I attempted to obtain the weight of ore passing through a tube-mill by taking the weight and specific gravity of the discharge from the feed cone, but could obtain no reasonably consistent figures. The ore contained quartz with a large amount of clay and oxide of iron; on drying it at steam heat and then determining its density in the usual way, quite discordant results were obtained, probably due to varying oxidation of the iron and dehydration of the alumina. Further attempts were made by taking a litre of pulp in a flask and weighing it, then allowing the solids to settle, and determining the specific gravity of the clear supernatant liquid. As much as possible of this liquid was then siphoned off

and the flask filled with a liquid of different specific gravity, such as alcohol or concentrated cyanide solution. After thorough mixing, the flask was again weighed, the solid allowed to settle, and the specific gravity of the supernatant liquid determined. From these weights values for the density of the solid were obtained, but these were scarcely better than those obtained by the drying method. Can Mr. Stadler help those in like difficulty who have not a nice clean ore of regular density to work with?

T. B. GREENFIELD.

El Oro, Mexico, February 8.

[The article on estimation of tonnage, by A. W. Allen, reprinted on page 454, will doubtless serve to meet our correspondent's need.—EDITOR.]

(March 23, 1912)

The Editor:

Sir—In your issue of February 24, T. B. Greenfield described at some length the practical difficulties met in determining the specific gravity of an ore, and in the same issue A. W. Allen describes methods which may be followed. Neither of these gentlemen allude to a factor which may easily be a source of error in such determinations, namely, the change in specific gravity of an ore due to the formation of colloid hydrates during fine grinding. According to the method of computation, this may be a source of error, or it may not. The mass (in the exact sense) of the ore is clearly not affected by chemical reactions, but its volume and, therefore, its specific gravity may be considerably altered. Where the solid material is separated, dried, and weighed the water of hydration would largely be lost, though changes due to oxidation may be of some effect, and the resultant error will be probably very small. But where, in the case of a slime, the method suggested by Mr. Allen on page 309 of your issue of February 24 is followed, x will represent the percentage of ore in the slime, but *not* the percentage of solid in the slime. Exact quantitative determinations of the influence of colloids on ore-dressing problems is much needed, and it is to be hoped that research along these lines may be undertaken in some of our technical schools.

THOMAS T. READ.

San Francisco, March 18.

RECENT CYANIDE PRACTICE BY DISTRICTS

METALLURGICAL PRACTICE IN WESTERN AUSTRALIA

By A. E. Drucker

(September 24, 1910)

There are at least nine noteworthy gold-producers at Kalgoorlie—the Golden Horse-shoe, Great Boulder, Ivanhoe, Perseverance, Kalgurli, Associated, Lake View, Oroya Brownhill, and South Kalgurli. Four of these crush their ore in Krupp ball-mills, one with both ball and Griffin mills, and the remaining four with stamps. The two main methods of ore treatment used at Kalgoorlie may be classified as follows: (1) Dry-crushing, all roasting, all-sliming in pans, and amalgamation, cyanide agitation of slime, and filter-pressing (the Great Boulder uses the Ridgway filter). (2) Wet-crushing in cyanide, classifying, concentrating, all-sliming in tube-mills, raw slime bromo-cyanide agitation (concentrate-roasting, amalgamation and all-sliming in pans, cyanide agitation), filter-pressing.

THE GOLDEN MILE, KALGOORLIE

Wet-Crushing with Stamps.—Of the various Kalgoorlie mines mentioned above, the Lake View, Oroya Brownhill, and the new 100-stamp mill at the Horse-Shoe are all-sliming. The coarse sand is ground fine in pans, and then slimed in tube-mills. (The Horse-Shoe employs tube-mills only for this purpose.) The Ivanhoe and the old 50-stamp mill at the Horse-Shoe treat the sand after a previous grinding in pans, by percolation, the slime being treated raw by bromo-cyanide agitation and filter-pressing. All these plants are wet-crushing in cyanide solution and have done away with the old plate-amalgamation. All amalgamating is now conducted in the grinding pans. Concentration is done on Wilfley tables, and the concentrate obtained at these plants is roasted before receiving cyanide treatment.

Dry-Crushing with Krupp or Griffin Mills.—The important dry-crushing plants of Kalgoorlie are the Great Boulder, Persever-

ance, Associated, Kalgurli, and South Kalgurli. These employ, mainly, ball-mills which are in high favor, and although they do not produce as fine a discharge, they are more economical in supervision, repairs, and power. The grinding-pan has played an important part in the metallurgical practice of Kalgoorlie. It has been used mainly with the all-roasting process in regrinding sand and amalgamating coarse gold. The gold won from this source varies from 25 to 35% of the total yield.

Telluride Gold Ores.—The refractory nature of the Kalgoorlie ores is due largely to the occurrence of the gold in combination with tellurium, forming a compound, $AuTe_2$ or $(Au,Ag)Te_2$, which is represented by the minerals calaverite and sylvanite. There is a difference in the ore at these various mines, due to variations in silica, sulphur content, proportions of telluride to free gold, occurrence of graphite, etc. The ore may be described as an altered greenstone schist containing from 50 to 70% of quartz. The min-

GOLDEN HORSE-SHOE GOLD MINE

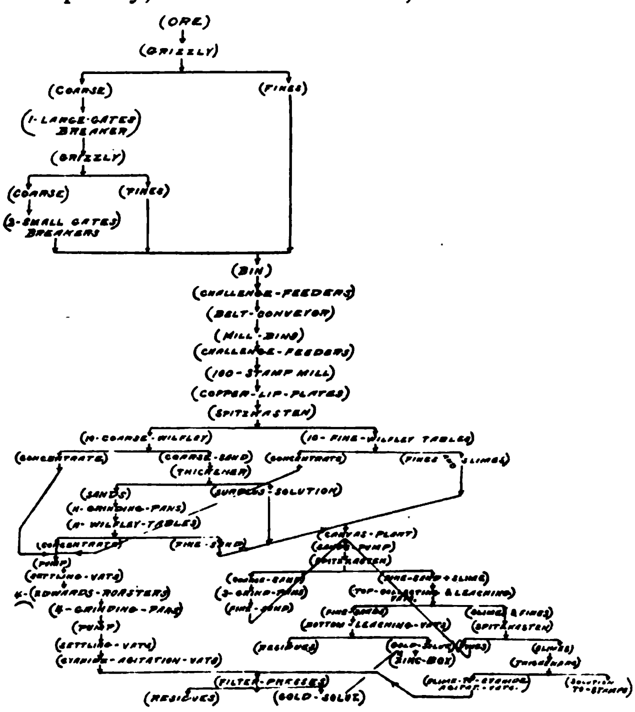
eral constituents are pyrite, pyrrhotite, and tellurides of gold, silver, and mercury.

Tellurides are widely distributed, and in some mines contribute a large portion of the total gold yield. Mines containing a notable amount of telluride are usually equipped with dry-crushing, all-roasting plants. Where tellurides occur in ores treated in wet-crushing plants, a careful concentration has to be resorted to previous to bromo-cyanide treatment. The concentrate collected is roasted before agitation with ordinary cyanide solution. Ordinary KCN or NaCN solutions are without solvent action on gold in combination with tellurium. Owing to the variations in the ore at the different mines each property had its own metallurgical problem to solve, and hence there is a great diversity of plants and methods. Kalgoorlie had not the opportunity to profit from experiences gained elsewhere in treating similar ores. It can be safely said that this district was the first to successfully solve the mill treatment of gold-telluride ores on a commercial scale.

The following analysis will represent the two different types of ore treated at Kalgoorlie:

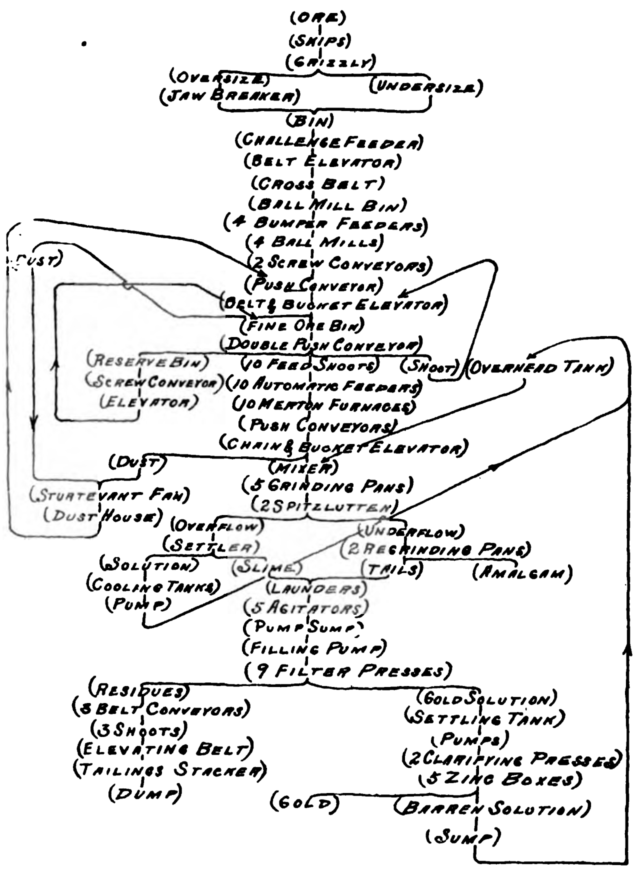
	Ivanhoe, Wet-crushing, Bromo-cyaniding. Per cent.	Associated, Dry-crushing, All-roasting. Per cent.
Silica	77.7	65.1
Ferric oxide	7.0	4.6
Sulphur	3.9	0.0
Alumina	1.5	2.4
Magnesia	3.4	5.5
Lime	5.4	6.7
Water	0.2	0.0
Undetermined	0.9	0.0
Iron pyrite	0.0	11.4
Alkalies, water, etc	0.0	4.2
	100.0	99.9

The Brownhill, Ivanhoe, and Lake View are all wet-crushing plants, while the Perseverance is dry-crushing, all-roasting. The wet-crushing process gives a net bullion extraction of 86 to 90% at a cost of about $2.30 per ton, while dry-crushing yields 90 to 94% of the gold with a cost of $2.70. It is generally conceded at Kalgoorlie that dry-crushing, all-roasting process has won, and is the most profitable method. However, I think that through increased stamp duty, screen classification, and closer concentration,



FLOW-SHEET AT THE IVANHOE

the wet-crushing process will prove to be the most profitable. In places where fuel and furnace supplies are high, the bromo-cyanide process would show an advantage no doubt. The chemical process



FLOW-SHEET AT THE SOUTH KALGURLI

requires considerable metallurgical skill and constant attention to the progress of each vat. The alkalinity of the cyanide solutions is important, the best extraction being at about 0.01% to nearly neutral.

CYANIDATION OF CRIPPLE CREEK ORES

By Philip Argall

(December 17, 1910)

*We are here tonight to help celebrate the successful starting of the second unit of Stratton's Independence mill. We thus reach an important point, not alone in the history of our own mine, but

*Address at banquet in celebration of the first month's run of the second unit of the new Stratton's Independence mill.

also in the history of the Cripple Creek district; the pioneer work is accomplished and wet-milling of $3 sulpho-telluride ores established as a profitable industry, away up here in the mountains where milling supplies are charged all the traffic will bear. Most of you have contributed, in one way or another, your knowledge, strength, and experience to help obtain this great result; though several of those who started out with us in the summer of 1907, in what was then called by hotel experts and club loungers 'the metallurgical impossibility,' have moved to other scenes and assumed new responsibilities while others again have but recently joined our ranks. To these younger men we look with expectant interest, for new ideas and for renewed energy which may lead to greater efficiency in all departments. At such a time as this one may be pardoned for pausing in retrospective mood to view again the point from which he journeyed and perhaps push the tentacles of thought out into the uncertain future, to see in vision, as it were, the ultimate metallurgical destiny of wet-milling the sulpho-telluride ores of Cripple Creek.

Eighteen years ago I first became interested in cyanidation and soon thereafter accepted the position of consulting engineer to the company holding the McArthur-Forrest patents for America. I made my initial trip to Deadwood, South Dakota, to examine into unexpected troubles that cropped up in the first cyanide mill built in the Black Hills, the ores of which were so favorable to cyanidation. The first mill erected was for a time a failure and but few, if any, foresaw the brilliant future for the cyanide process in that great mining district. My second trip was to Cripple Creek, where a small cyanide plant had been erected, later called the Brodie mill. This mill failed to give the results expected, as had also the one at Deadwood, and for precisely the same reason. The ore after grinding could not be leached. Improved crushing machinery, however, solved the problem in both cases, and the Brodie mill struggled along for some time at a capacity of 15 tons per day, later raised to 30 or 50, and ultimately, I believe, to 100 tons per day. In the spring of 1894 we had no difficulty in procuring a full supply of ore for the Brodie mill of about one ounce value per ton, for which the mill received $15 per ton treatment charge and needed every cent of it. My connection with this mill, though short, was ample to convince me that cyanidation had a great future in the metallurgy of Cripple Creek ores. I consequently experimented quite extensively with the telluride ores and, in fact, worked out, wrote up, and published the identical method of treatment now in use at Stratton's Independence mill. I proclaimed the cyanide process to be the most suitable all-round method for treating Cripple Creek ores, a thesis I stoutly maintained with tongue and pen against all comers, until the use of cyanide became universal in the milling of Cripple Creek ores.

The fall of 1904 found me engaged in building the first large custom mill for the direct cyanidation of telluride ores, while the following year I introduced roasting as an important step in cyanid-

ing sulpho-tellurides. The Metallic works, also a pioneer in Cripple Creek metallurgy, ultimately reached a capacity of 10,000 tons per month, and at the close of my engagement, January, 1901, had treated almost half a million tons of Cripple Creek ore, mostly by the roasting process, and had from the first earned good dividends on the investment, and this in the face of a steadily decreasing treatment charge.

From my first connection with Cripple Creek milling, to the close of 1900, the average treatment charge had been reduced 50 per cent, and it must be conceded that the works of the Metallic Extraction Co. was an important factor in this great reduction. Were I relating a personal narrative, or holding forth on my varied experience in cyanidation, I would next direct your attention to Mexico, to Canada, and to other countries. I am, however, merely tracing the progress in cyaniding Cripple Creek ores and incidentally, though briefly, noting my own pioneer work in that connection. Suffice it to say, then, I returned to this field of activity in 1906 and early the following year took up the greatest ore-treatment problem of my life. To understand it clearly, it might be permissible to say that at the Metallic Extraction Co.'s works near Florence, where fuel and general supplies were reasonable, the climate mild, and water abundant, I had gotten the ore treatment cost down to what I then considered a low figure. The problem involved in Stratton's Independence dump, however, contemplated the profitable treatment of an ore—including mining it in the dump and conveying it to the mill—the total value of which was less than our average cost for treating a ton of ore in the Metallic works in the year of grace 1900. Here, then, was a problem of some magnitude and I will frankly admit that it was only the quantity available in the dump, something like a million tons, that induced me to make the attempt. Roasting was out of the question on account of the cost; so I went back to my old concentration tests of 1894, and found that modern concentrators and fine grinding gave very encouraging results. A long series of experiments proved to my satisfaction that 35% of the gold value could be removed as concentrate from average dump ore, and, strange to say, cyanide was good for a similar percentage. Next came the cost of the method, to determine which I had to draw on previous experience in concentrating and cyaniding on a large scale. Finally, having proved my experimental work in all particulars, I cabled the company in March, 1907, that a mill of 10,000 tons monthly capacity could treat the dump ore by the proposed method at a cost of $1.50 per ton, obtaining a yield of 70% of the contained gold. Those figures, as you know, have been exceeded in actual milling results, and still higher extraction is attainable by finer grinding, but with the present cost of power and supplies is scarcely justified from a commercial point of view.

The method used in our mill—we have never called it a process—is but a combination of well-known devices and chemicals to obtain the desired end; we have no secrets, chemical or otherwise,

and from the first day the plant has been open to the inspection of metallurgists and all information or data asked for by responsible parties has been frankly supplied. The stride from a $15 per ton milling cost in 1894 to a cost of $1.50 per ton in 1910 is a great one. Still, I believe the milling of low-grade sulpho-telluride ore is today in its infancy; improvements in machinery and methods will come, making toward higher efficiency, better extraction, and lower working cost. For the straight wet-milling of sulpho-telluride ores a $1.25 working cost is now in sight on a basis of treating 10,000 tons per month, while the dollar milling cost is perhaps not far distant, and could possibly be attained in a well designed plant treating not less than 15,000 tons per month. In neither estimate, however, is amortization taken into account; the figures cover only the bare milling and maintenance cost.

Stratton's Independence mill, delayed for a time through financial reasons, started in April, 1909, with a nominal capacity of 4500 tons per month, enlarged in December of that year to 7000 tons, and with the addition of the complete second unit last month, reached a capacity of 9000 tons. The mill, through the energy and ability of the staff, was profitably operated from the start, and is now earning 10% per annum on the capital of the company, and has treated to date 120,000 tons of dump rock; sufficient, I believe, to take it out of the class of 'metallurgical impossibilities,' if not to establish it as one of the chief industries of the Cripple Creek district. We all, more or less, realize what the local milling of the low-grade ores means to Cripple Creek. It does not offer a large or immediate reward to the mine-owner, but on the contrary calls for a considerable expenditure of capital; hence the development of the milling of low-grade ore must proceed conservatively along the lines of consolidation of small properties or joint milling on a co-operative basis. Milling the low-grade ores in the district does, however, mean the maintaining of, and possibly an increase in, the output of shipping ore; the prolongation of the life of the camp, I might say, indefinitely; the steady employment of large numbers of men in the mines and mills and the purchase of vast quantities of supplies. In a word, high-grade production tends to make millionaires, the low-grade, a populous, prosperous, and permanent community. The Cripple Creek district will, I hope, continue to give us an occasional millionaire and an ever-increasing community of active and prosperous workers.

I believe, gentlemen, you will ever find cause for congratulation in the fact that you were pioneers in this low-grade milling industry. Your work has shown that sulpho-telluride can be concentrated, that the tailing from the concentrator can be cyanided, and ores assaying less than $3 per ton can be milled at a profit; and I sincerely thank you for your co-operation and assistance in this great work, second to nothing that has ever been undertaken in the Cripple Creek district; where you blazed the way others can safely follow, successfully copy, and, let us hope, in time improve. Nevertheless, I would not have you forget that great as has been our

responsibility, and arduous the work, yet I believe the ultimate credit is due to the directors of Stratton's Independence, Limited, who eight years ago, realizing that one-fifth of the production of their mine was finding it way into the ore-house dump, started research work looking toward its recovery; first, by electro-cyanide methods, later by erecting and operating an experimental plant at a cost of nearly $60,000, and experimenting in various ways for fully four years; lastly, sanctioning the expenditure of more than a quarter of a million dollars in the plant we erected and are now operating. At all times I felt I had in our board real men behind me, gentlemen who supported and encouraged me at every step and who never lost confidence in the ultimate outcome of the undertaking. To these gentlemen is due, then, the full credit for introducing low-grade milling into the Cripple Creek district, and it is through their keen business acumen that success has been achieved and that we are here tonight to celebrate it, even at a time when others continue to use their low-grade ore to ballast railways and fill waste places along the right of way. Much has been said and written of late about the great and little-understood doctrine of conservation, but here is true conservation, the creation of a great and profitable industry from the waste rock of yesterday. A new era is dawning over the Cripple Creek district, local milling is firmly established by two great mills, the largest ever erected in the district. This, however, is but the beginning of home treatment, which, I believe, will rapidly expand in the near future and very soon cover the entire field. Cyanogen is king.

AMERICAN PROGRESS IN CYANIDATION

By An Occasional Contributor

(January 7, 1911)

Each year sees a marked increase in the percentage of the world's precious metals recovered through the agency of cyanogen. In the early days of the cyanide process, the opinion was prevalent among metallurgists that the successful use of cyanide would be confined to a few special instances where all other means had proved to be futile and where the physical and chemical natures of the ore were particularly favorable. Today practically the first questions asked when attacking the problem of treating gold-silver ore, are, what extraction can be attained by cyanide, and the cost of making it. It is only after these conditions are shown to be unfavorable that the possibilities of other processes are considered. As in the history of most of the other arts, the first years of cyanidation saw improvement follow improvement in such quick succession that there was great trouble in keeping pace with the march of progress, but as time went on, these improvements became less spectacular until today changes are more refinements of existing features than anything positively new. There are now plants that have attained a remarkably high degree of metallurgical perfection.

As has been the case for several years past, Mexico has led this year, though mostly through the efforts and ability of the Anglo-Saxon engineers. Several remarkable new mills have been put into service there during the past twelve months, and several more are to follow in the first months of 1911. The flow-sheets of these new mills show that the 'Pachuca tank' will be used in almost every one. It is rather curious that although this was the invention of F. C. Brown, of New Zealand, it is used scarcely at all in Australia. A striking innovation has been made by the introduction of the continuous system of agitating which has proved an unqualified success. In the old method the pulp was agitated in separate vats which had no communication with each other, each constituting a self-contained unit. By the continuous system the 'Pachucas' are arranged in series or in batteries, and the pulp overflows from one into the next and passes through the whole series before its discharge to the filters. The method of communication used is a pipe set at an angle of 60°. The upper end of the pipe is placed in the discharging tank, half way between the central airlift and the side of the tank, while the lower end is placed in the same relative position in the receiving tank. The upper end of the pipe is distant about one-third of the height of the tank from the top, while the lower end is one-third from the bottom. A. Grothe states that both M. H. Kuryla and J. E. Mennell report increased extraction as a result of using this method, but this would seem probably to be due more to the fact that the old system of agitation was not carried to the point of maximum dissolution, than to any other reason. There is no reason to doubt, however, that there is a considerable saving of cyanide where the continuous process is used. Other advantages are, that the tank is used continuously for agitation, and no time is lost in filling and discharging. Also, the discharge being from the side, and not from the bottom, there is an economy of gravity pressure, which in most plants is important. Furthermore, the discharge being continuous, the pressure is constant; which is essential in many cases where the pulp is filtered in pressure-filters.

The use of the silica-sponge diaphragm has caused a good deal of discussion during the year, but so far has not proved of great value. The idea, as well as the application, is so new that further service may suggest changes that will bring it into more extended use. The reported disadvantage is that the volume of air which can be passed through the sponge is too small to prevent classification in the pulp, and that the interstitial spaces become choked easily and are hard to clean. Other operators report very successful results. C. A. Fullon says that in the San Matias mill at Guanajuato the process has been very successful and has reduced the time of agitation from 32 to 12 hours. It would seem that for successful use, the pulp agitated should be ground very fine and contain a very high percentage of floculent. The Clancy process was announced late in the fall, but up to the present it has not been tried in enough places to give sure basis for judgment as to its practical success. At one well known property the experiments were brought to

rather an abrupt termination at the instance of the management, as experimental work lasting over a considerable length of time proved that, so far as this particular property was concerned, there was little hope of its being used to advantage. Further application of the process will be watched closely, as the problem of treating ores containing combined gold is one of much interest. The wonderful increase in the use of the Dorr thickeners and classifiers shows that they have come to stay, for a time at any rate, in slime plants. Some new things are promised soon from South Africa along these lines in addition to the Caldecott diaphragm-cone, but a discussion of these will come better from the pen of some South African.

The various forms of filters, both vacuum and pressure, are still merrily warring for the favor of the profession. Among the former the Butters and the Oliver are going into many of the mills that are being designed by engineers favoring vacuum-filters. The Butters company is providing a filter of 1000-ton capacity for the Dos Estrellas which will handle the slime from the No. 1 and No. 2 plants, and it is understood that the Real del Monte is to use the Butters in its enlarged mill. Another new Butters installation is at the Palmilla, at Parral, while the La Blanca and others have put in Moore filters.

Among the pressure-filters the Merrill seems to be gaining great headway. The Santa Gertrudis engineers decided on this type for their new 600-ton mill after competitive tests (held at the mine on ore of the usual character), and in which a number of manufacturers of different types competed. The Merrill filters, it is also given out, are doing good work at Esperanza, where the management has recently given an order for one more, making six in all. The New York Honduras & Rosario Co. is also installing two of the large 90-leaf type in Honduras. At first when the Merrill press was used it was found that a certain amount of granular material was necessary in the slime. By a very ingenious idea, known as 'centre washing,' this objection has been largely overcome. The 'centre washing' system consists simply of filling the press to a point where there is left an opening through the centre of the cake about ¼-in. wide. Solution or wash-water is introduced into this opening through the nozzles on the automatic sluicing-bar or through the sludge-feed channel, and leaching begins. The flow of liquid commences at the centre and proceeds outward in both directions. The result is that instead of a solid 4-in. cake (which if very talcose, was formerly impermeable), there is now in the same leaf two cakes, each 1⅞ in. thick, which are readily permeable, even when formed of the most talcose slime.

Among other filters, the Kelly seems to be achieving an increasing measure of popular favor. The great value of the Kelly lies in the small amount of water used in its operation, which is no small factor in many places where the water supply is limited and evaporation heavy. At the new mill of the Veta Colorado, at Parral, 8

Kelly presses working in 2 batteries of 4 each, are being installed, and 2 are in use in the Carmen mill in Guanajuato. The Tigre mill in Sonora is also to be equipped with Kelly filters. This last mill is rather remarkable among Mexican plants from the fact that its stamps are to weigh 1250 lb. It would rather seem that there is a tendency toward heavier stamps in Mexico as well as elsewhere. Certainly the trend is all that way farther north, although as yet designers in the United States have not dared to go as far in the matter as have engineers in South Africa, where some of the latest plants include stamps of 2000 lb. weight. A number of mills are being designed for work on gold ores containing stamps weighing 1250 lb., and the probabilities are that, if these show good economic results, the fashionable stamp-mills of next year will include still heavier heads. Tube-milling is also gaining in favor steadily, as it has been shown that fine grinding results almost invariably in increased amalgamation as well as added extraction in the subsequent cyanidation. The distinctive feature, however, of the fine grinding of gold ore seems to be in a release of the gold in the sulphides, thereby shortening the necessary time of contact with cyanide solution. This makes possible in many cases the direct cyanidation of the sulphides in the same flow with the slime and thereby does away with losses due to concentration.

At Cripple Creek the subject of the economical handling of the low-grade sulpho-telluride ores remains still the important one. The metallurgists of this district have worked untiringly on this problem, and this year brings high hopes that the solution is near at hand. At the Stratton's Independence a second mill-unit has been started by Philip Argall. The process used here was described by P. H. Argall last year.* In the middle of the summer the new mill of the Portland G. M. Co. was started. The company has been loath to give out information as to results until success had been demonstrated beyond doubt, hence the incompleteness of published descriptions of the new mill. In a general way, it is understood that low-grade sulpho-telluride ore assaying $4 to $5 per ton is slimed by standard methods and agitated for about six hours with cyanide solution in Pachuca vats. This is followed by another agitation with bromo-cyanide and the pulp filtered and washed. It is said the extraction is about 80%, and the cost reasonably low, considering the complex nature of the ore.

In Nevada the year shows the Montgomery-Shoshone mill shut down, probably for good, but on the other hand it now seems certain that in the spring work on two new mills will begin. The Belmont will have its own plant, situated at Tonopah, and in all probability this mill will be modeled along the same lines as the Montana-Tonopah, which continues to give good satisfaction, and in which there has been a steady improvement in methods from the first. The Nevada Hills will, in all probability, erect a mill in the spring, although the tonnage to be treated still seems to be in doubt.

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It is rumored that the plant will contain 60 stamps and have a capacity of 500 tons per day. Being controlled by the same interests as the Goldfield Consolidated, it is not surprising that the provisional flow-sheet shows similarities to that of the great Goldfield mill. No article of last year's cyanide work is complete without some mention of the fire at the latter and the remarkable rapidity with which operations were resumed. It speaks volumes for the pluck and energy of the men responsible that, within a few weeks after such a disastrous fire, the mill was running as smoothly as ever. The Goldfield Con. engineers deserve the thanks of the profession for the information contained in a very attractive booklet which was sent to a large number of professional men. The information contained will be of great value, owing to the graphic representation of the actual operating factors in a large and modern plant. J. R. Finlay and J. W. Hutchinson are to be congratulated on having taken a step which it is hoped others will follow. Reproductions of three of these charts, covering filter-discharging, crushing-conveying, and tube-milling costs, accompany this article. In examining them it should be remembered that during April, May, June, and July only 70 stamps were dropping. Not only are the costs high for these months, but the averages for the year were necessarily raised by this circumstance. At the Pittsburg Silver Peak mill, 20 additional stamps have been put into service during the year, bringing the total up to 120. About 15,000 tons of ore per month is being crushed. Considering the fact that this company is operating in Nevada and under typical Nevada conditions of high cost for labor, power, water, and supplies, the working costs are little short of marvelous. The latest figures given out are $2.54 per ton, which includes not only the cost of the Pittsburg office, but takes care of all payments made for improvements and additions.

In California the new mills of note during the past year are those of the Trinity Gold Mining Co., in Trinity county, and the recently started cyanide plant of the Empire mine at Grass Valley. The Trinity mill is complete and modern, and should do good work. The process consists of crushing (40 stamps), amalgamation, separation of the pulp into sand and slime by Dorr classifiers, leaching of sand in vats, agitation of slime in 'Pachucas,' followed by Dorr thickeners, Oliver filters, and Merrill precipitation presses. The cyanide annex that has just been added to the Empire's crushing and amalgamation plant contains Oliver filters for the slime and Merrill precipitation presses. This plant displaced the old-fashioned concentrating and canvas plant which has for many years done such good work. At the Homestake in South Dakota, there have been no changes of note during the past year. The slime-plant, with its 30 Merrill presses, has now reached a point where a 90c. product is being treated at a cost of 20c. per ton.

In Washington State a large mill will be erected in the spring near Republic, where mining is becoming more active. The metallurgy of the Republic ores is extremely difficult, and, as the ore

is of low grade, large tonnages must be handled in order to make a profit. Although the new mill is not yet under way, it is understood that it will consist of 100 heavy stamps followed by fine-grinding, probably in tube-mills. Agitation in Pachuca vats will be followed by Butters filters and zinc dust precipitation. Engineers of the Mother Lode company are at present busy with the designs for a plant to be erected near Salmo, and work on the first unit will be started in the spring. In Ontario, spring saw the commencement of operations in the Nova Scotia mill at Cobalt, which was the handiwork of A. G. Kirby, of Reno, Nevada. The results obtained are said to be excellent, and this being the case, it would seem as if the main features of the milling of Cobalt ores were settled. Of one thing there can be no doubt, and that is, that Cobalt in time will come to be a district of many mills. The probability is that the high-grade ore will always be shipped, but operators will find not only the low grade, but probably the middle-grade ores to be well adapted to treatment in well designed mills. A short distance from Cobalt is the newly discovered camp of Porcupine, concerning which there is only one feeling on the part of the men who own ground there, and that is one of supreme confidence. This is being expressed by putting up mills. Two are now certain, with rumors of several more. On the Timmins property, a big mill will be built, but work has not yet begun. On the Dome property the design of the plant has been completed, the machinery ordered, and the coming spring will see a large force of men at work. The Dome mill will consist of 40 stamps of 1250 lb. weight, followed by amalgamation, Dorr classifiers and thickeners, tube-milling, agitation in Pachuca vats, Merrill slime-filters and Merrill zinc dust precipitation.

Some work is being done in Alaska, but the only feature to be commented on there is the announcement of operations in the cyanide plant of the Alaska Treadwell, where a mill has been built for crushing and cyaniding and the concentrate that in the past has been shipped and smelted. At the termination of the present smelting contract, February 4, this mill is to be put in regular operation. The most striking peculiarity is that 80% of the gold in the concentrate will be saved on plates following tube-milling and prior to cyanidation. The new mill will effect a great saving for the company, and R. A. Kinzie and his associates deserve great credit for their work.

Summing up now briefly the features of the year's work in North America, the following stand out prominently: (1) The increased use of Pachuca vats and the application of the process of continuous agitation in them. (2) The tendency to demand heavier crushing duty of the various crushing machines, particularly of stamps. (3) The continued demand for fine crushing with attendant changes in classification and filtering practice. (4) The increasing growth of the practice of cyaniding the sulphides and vein-material in the same operation.

CRIPPLE CREEK METALLURGY

By Thomas T. Read

(February 18, 1911)

The progress made toward solving the problem of how best to treat the rather difficult ores of the Cripple Creek district is perhaps more striking to one re-visiting the camp after four years absence than to those who have watched its daily growth. In any case the problem of ore-treatment at Cripple Creek possesses so much interest that no excuse is required for adding to its discussion.

The unoxidized ore at Cripple Creek consists essentially of tellurides of gold in a gangue of which the most significant feature is its high aluminum content. These give us the governing conditions; the gold can not be recovered by amalgamation or cyanidation in the raw state, the ore can not profitably be concentrated because of the brittleness of the gold mineral, and can not be smelted except when mixed with a large quantity of easier-smelting ore. Roasting is expensive and produces small lumps of metallic gold that are difficult to catch by amalgamation and impossible to dissolve in cyanide solution in any reasonable length of time. Some years ago the problem had been brought to this stage of adjustment; the rich ores were sacked and sent to the smelters at Pueblo, Denver, and Salida (where some five years ago the smeltermen found that Cripple Creek ore could be used to advantage to replace the lime formerly used in the pot-roasting of lead ores to keep the charge from fusing in the pots). Only the richer ores can be treated thus, as the treatment and freight charges become too severe a tax on the lower-grade material. That of medium grade was sent to mills at Colorado City where it was first roasted and then chlorinated. The low-grade ore was left in the stopes or piled on the dumps.

Meanwhile Cripple Creek was fast approaching that stage in the development of every mining camp where the former medium-grade ore becomes high-grade and the former waste is regarded as low-grade ore, the change being due both to the exhaustion of the richer ore and the lowering of the working casts. From time to time mills were started adjacent to the mines with the intention of treating lower-grade unoxidized material than could be shipped to Colorado City. These were generally unsuccessful, for a variety of causes.

About four years ago the opinion had become general that the chlorination process, preceded by roasting, was unsatisfactory because of the high gold content of the tailing and the high cost of treatment. Two improvements were made; the chlorination process was supplemented by concentration and by cyaniding the tailing; and a new process of roasting, re-grinding, running over blanket strakes to catch the free gold, followed by cyanidation of the ore.

In the face of the decreasing grade of the average ore, the items of $1 for freight to Colorado City and $0.70 (about) for roasting became painfully significant. It was clear that some method must be found of treating the ore near the mines, without roasting.

The problem of treating the unoxidized ore without roasting hinges on the fact that the gold telluride is not soluble in ordinary cyanide solution. It can be made to dissolve by using a solution that has great oxidizing power. One of the first methods tried was bromo-cyanidation. There have been no published accounts of the detailed results of experiment on these ores, but the impression is general that bromo-cyanidation fell before the crushing blow of high working costs, coinciding in a general way with Western Australian experience in this regard. It is not dead, however, as it is used in certain instances to supplement the main method of treatment, and I regard it as not improbable that with further experiment bromo-cyaniding may be found useful in many cases. The fact that the whole history of the cyanide process has been a constant struggle to perfect the mechanical equipment used must never be lost sight of in considering the merits of any proposed process.

A significant feature of progress in the past year has been the introduction of the Clancy process for treating unroasted ore. Its essential features are the use of cyanamide and iodide in the solution and the use of the electric current for producing a high degree of oxidation without excessive consumption of cyanide. The impression among the best informed is that the process obtains the desired extraction, though no actual working costs have been made public. A great deal of the discussion of this process has been footless and entirely beside the point. Metallurgy is chemistry applied to business, and if by treating gold ore with a solution of old boots a good extraction at a low cost of operation can be obtained, it is good metallurgy; and if by treating with C. P. chemicals in apparatus of the latest refinement of design the extraction is poor and the cost high, it is poor metallurgy. When the Clancy process demonstrates low costs of operation its position will be impregnable.

The present stage of development in treating the raw ore is to first crush incyanide solution and classify the product into sand and slime, using either the Dorr or Akins classifier. The sand is concentrated on some approved type of table, the tailing rejected and the concentrate sent with the medium or high-grade ore to Colorado City or Pueblo. The slime is treated by some process that will give the necessary oxidation. The most important mills are not yet ready to present their processes for criticism and comparison, but by the end of another year it is hoped that a more detailed discussion of progress can be made.

Meanwhile we have the brutal fact to face that the shipping ore from Cripple Creek is steadily decreasing in grade. This is partly offset by the fact that the constant pressure on the Colorado smelters to obtain silicious ores to smelt with the flood of basic

ores produced in most of the Colorado camps makes them more willing to take Cripple Creek ore at a low treatment charge, even though it is aluminous rather than silicious.

TREATMENT OF SILVER ORES AT GUANACEVI, MEXICO

By R. C. Kline

(March 18, 1911)

The Guanacevi district, situated in the northwestern corner of the State of Durango, has for many years borne an important part in Mexico's production of silver and its yearly output is still not inconsiderable. Owing to the comparative inaccessibility of the camp, lack of cheap power, and highly refractory character of most of the ores, little progress has been made in handling the ores of low value. There is now a strong probability that a line of railroad will shortly be built in from Tepehuanes, and if this be done the situation should be very much improved.

Upon my arrival at Guanacevi in the spring of 1907, I found that none of the companies in operation at that time were successfully cyaniding their ores. One company was getting ready to build a new cyanide annex to its concentrating mill, while another was still trying to get a satisfactory re-arrangement of their equipment and was putting in some of the more modern devices at the disposal of the metallurgist. Former attempts to treat by coarse crushing, with separation, leaching, and air agitation in flat-bottom tanks, had proved complete failures. This work had been done before much of anything was known regarding the cyaniding of silver ores, and the results in no way reflect discredit upon the men who were responsible for them.

The first company for which I examined ores was developing a portion of the Santa Cruz vein. The ores of this vein may be divided, for purposes of this article, into two classes: oxidized and unoxidized. It is concerning the oxidized ores that I shall write. These apparently varied little in their metallurgical composition, but yielded vastly different percentages of extraction under identical conditions of treatment. Oxides of iron and manganese were present in considerable quantity, together with small amounts of zinc. The ores nearer the surface showed less zinc than those from deeper levels, but contained approximately as much manganese, in the form of pyrolusite, as did the others. This fact is interesting, since the presence of pyrolusite in considerable quantity is believed by many to be the cause of poor extraction; yet the ores from the upper level of the property in question yielded over 90% of their silver without any special difficulty. A curious point in this connection was the fact that the ores from the second level yielded a much poorer extraction; those from the third level also poorer, but better than those from the second; while those from the fourth level gave a good extraction, con-

sidering the nature of the ore, in fact almost equal to those from the first. Black oxide of manganese was present in practically the same amount in all four samples, and its presence could have, therefore, little bearing upon ultimate results. Nor would it appear probable that the varying extractions obtained were due to intermediate oxidation products of the manganese minerals, even though many of the unoxidized ores of the district show heavy percentages of pink rhodochrosite and rhodonite.

Work on the above ores was begun by crushing in solution to a total product, or 'slime,' all of which would pass 200 mesh. The crushing dilution was about 8 to 1. Agitation was carried out for varying periods and with different dilutions. The resulting extraction of gold was excellent, and although the gold value of the heading was low, in no case did I find over 8c. per ton in the residues, while they more often assayed as low as 5c. This figure is perfectly trustworthy, as from 40 to 50 buttons from 1 A. T. charges were always combined and parted for a gold weighing. The use of chemical agents, such as lead acetate, barium peroxide, mercuric chloride, ammonium chloride, and bleaching powder, was thoroughly tried, but in no instance with any appreciable benefit. The effect of varying solutions showed very clearly that anything greater than three of liquid to one of solids was of absolutely no benefit, with the possible exception of a very slight cut in the time required for completing solution of silver. This was not sufficient to balance the increased cost of equipment and power, for the larger quantity of material. This is strictly in line with later experience of others, and of my own. The increase in time of agitation, from 24 to 30 hours, in every case showed that the added cost for power and tankage was more than repaid and the results were consistent. Periods of agitation greater than 30 hours did not, however, show any material or commercial profitable increase in extraction of the silver. Concentration tests, carefully made by passing the material over a small canvas table, showed in every case that after treatment had been completed, the concentrate did not contain enough to pay costs of concentration, freight, and treatment charges. This matter was thoroughly checked and concentration, as a commercially profitable means toward treating this class of ores, was eliminated.

It will be noted that my method for deciding the advisability of concentrating an ore, is to extract the concentrate from a sample of the ore which has already been subjected to cyanide treatment. This, in combination with a narrow and long canvas table, which insures catching the fine material, gives conclusive evidence one way or the other and can be afterward checked by tests on fresh portions of the sample containing the concentrate and others free from sulphides. The latter tests will show the additional consumption of chemicals, if there be any. At the time this work was carried out, Charles Butters had comparatively recently presented to the metallurgical world his admirably effective vacuum-filter and most millmen believed in giving at least one extra wash

and decantation to a slime charge before sending it to the filter. More often they gave three or four. We were pretty well fixed in our ideas on this matter, at least, those of us who had had previous experience with the old type of filter-press, in which, owing to the segregation of the crystalline portion of the charge in the lower part of the cake frame, complete washing of the colloidal, or upper portion, was impossible. E. M. Hamilton proposed and adopted the modern slime treatment, whereby the charge, after having been subjected to agitation until solution of metals was commercially complete, was settled, decanted, and the thickened pulp run direct to the filter, there to be washed practically free of soluble metals. This mode of procedure he had already adopted at the numerous mills of the Charles Butters company.

I tried this method on the ore in question, carefully checked it against identical samples treated by one and by several extra washes and decantations and found that washing was practically complete on the experimental filter. In view of later practical experience of my own and of many others, these views are of course proved correct. Leaching tests on samples carefully classified wet and free from any considerable amount of colloidal slime, were made. This work, while carried out on a small scale, was entirely practical in its results, care being taken to have a leaching depth of sand equivalent to that of practical plant work, thereby giving the same conditions of packing of charge, length of column of sand through which the solution must pass, etc. Preliminary leaching tests showed that in order to obtain the best commercial result the sand must come fairly close to passing 100 mesh.

The final results were good, but when treatment was complete, the coarser material sized out, recrushed through 200 mesh and agitated, sufficient additional recovery of silver was shown to pay for the power and chemicals and to leave a small profit beside. Crushing through 200, or 'total sliming,' was therefore decided upon. An attempt to improve extraction by first giving the ore a sweet roast gave astonishing results, and instead of improving matters it lowered the silver extraction to a little over 10%. It would be interesting to hear of the experience of others in this connection.

The question of coarse free gold was effectively settled by a series of amalgamation tests. The crushed ore was made up to a thick pulp and, having sufficient mercury present, was thoroughly agitated in bottles. The pulp was subsequently thinned, washed free of mercury, and after dewatering, was given the regular treatment. Parallel samples were given identical treatment, but without attempting to amalgamate any gold. The results in both series tests averaged practically the same extraction. The treatment decided upon for this class of ore was therefore essentially as follows:

1. Crushing in solution, so that no more than 2% would remain on 200 mesh.
2. When a complete charge had been milled into a tank, to add sufficient cyanide to bring the solution up to a strength of

2.5 lb. true free cyanide per ton, as determined by silver nitrate, without the iodide indicator or the addition of any caustic alkali other than that already in solution.

3. Agitation to be carried on continuously for not less than 30 hours. After decantation of the resulting solution, the charge to be run to the filter:

4. Filter-cakes to be washed, in position, with from 1½ to 2 tons of barren solution per ton of slime, and then discharged from the filter.

5. Lime in sufficient quantity to maintain a protective alkalinity of 2.5 lb. per ton of solution in terms of sodium hydrate, to be added at the battery. In view of later practical experience with other classes of Guanacevi ores, this figure was proved to be too high, the best strength being about 2 lb. per ton or slightly under.

The unoxidized ores of the district presented a much more varied mineralogical character and a cursory description of some of the combinations encountered may be of interest. They are: (1) A clean quartz, in which the silver, as evidenced by high-grade specimens, appeared to be chiefly in the form of the sulphide, with smaller quantities of double antimonial sulphide, or stephanite. Little or no light or dark ruby silver in evidence. This ore treated easily to about 89% silver extraction. (2) A very clean ore, with little mineralization apart from the silver, small amounts of which were in the form of dark ruby, or pyrargyrite. Samples proved streaky, with regard to treatment, but the reason for this was not ascertained. Ores with gold content up to $28 per ton easily yielded over 96% extraction, with an average silver recovery of about 80% by straight cyanide. (3) A light brown ore, evidently partly oxidized. Treated to about 84% silver and over 95% of the gold. (4) A hard and heavily mineralized quartz, containing considerable quantities of pyrite and marcasite, with less galena, sphalerite, rhodochrosite, and rhodonite. It proved refractory, and an examination of high-grade specimens showed that the silver was chiefly present as argenite and stephanite, with smaller quantities of ruby silver. It yielded about 77% of its silver and 93% of its gold. (5) Ore from the same mine as No. 4 and essentially the same in character except as regards the quantity of sulphides present. It proved to be the most refractory found and yielded only 50% of its silver. About 24 hours continuous agitation was found to be sufficient for the solution of all silver capable of being removed by cyanide treatment and, as in the case of the oxidized ores, the use of other chemical agents gave no additional extraction whatever.

In view of the fact that the silver proved refractory while the gold responded easily to treatment, in the sample, an examination of the probable chemical reactions involved in the case of the silver is of interest. This matter has already been presented in a very interesting manner by Francis J. Hobson, in a paper contributed to the *Mining and Scientific Press* some time past. I regret that I have no copy of his contribution at hand, and it must be at the

risk of repeating what he may already have said in this connection, that I go into the subject.

Probably no positive knowledge of the more complicated silver reactions is available. Regarding the more simple silver combinations, it is known that: (1) Argentite and cerargyrite readily go into solution, forming double silver potassic cyanide and the corresponding alkaline sulphide and chloride, respectively; (2) that the silver thus dissolved will stay in solution in this form provided there be present sufficient zinc or lead salts to precipitate the alkaline sulphide. In treating many ores and tailings, there is enough zinc dissolved to answer the purpose, without resorting to the use of lead acetate or litharge.

With stephanite and ruby silver the case is somewhat different. It is known that these minerals are attacked by potassium hydrate, which turns them black and under favorable conditions partly dissolves them. What probably happens may best be seen by examing the following reactions. The chemical composition of the minerals in question is:

Stephanite $(2Ag_5SbS_4)$——$5Ag_2S.Sb_2S_3$
Pyrargyrite $(2Ag_3SbS_3)$——$3Ag_2S.Sb_2S_3$
Proustite $(2Ag_3AsS_3)$——$3Ag_2S.As_2S_3$

Under the action of potassium hydrate, stephanite, for example, would probably react as follows:

$$2Sb_2S_3 + 10KOH = 4KSbO_2 + 6KSH + 2H_2O$$

The potassium hydrosulphide thus formed would react on another portion of antimony trisulphide, thus:

$$3Sb_2S_3 + 6KSH = 6KSbS_2 + 3H_2S$$

The hydrogen sulphide formed in the above reaction might, in its nascent state, further interact with potassium hydroxide, forming nascent potassium sulphide, which would react on another portion of antimony trisulphide thus:

$$Sb_2S_3 + 3K_2S = 2K_3SbS_3$$

Of the above reactions, the first two, resulting in the formation of potassium metantimonite and metasulphantimonite, appear to me to be the most probable. It is, however equally possible that some modification of the third, resulting in the formation of potassium sulphantimonite and potassium sulpharsenite, where proustite is involved, takes place. In any event the sulpho compounds are strong reducing agents, tending to take up oxygen and go to the mono and poly sulphoxy arsenates and antimonates. This fact takes on added importance if there be gold or native silver present in the ore.

That the action of potassium hydrate is highly beneficial in breaking down the above silver combinations, thoroughly proved by experimental work done by myself on a sample of high-grade

silver ore carrying over 1200 oz. and on which ordinary treatment with potassium cyanide solution carrying enough cyanide to completely dissolve the silver, together with sufficient lead acetate to protect against re-precipitation, only served to extract some 40%. On subsequently boiling with a solution of caustic potash and resubmitting to the action of cyanide solution, sufficient additional silver was dissolved to bring the extraction up to about 90%. But while this method of recovery was interesting, from the chemical point of view, throwing some light upon the benefit to be derived from the use of strong caustic alkalies, it is unavailable in the case of practical operations, owing to the impracticability of heating up large quantities of pulp. Nor does it give uniformly good results in all cases.

The use of strong caustic soda solution failed to give satisfactory results. It is probable, therefore, that potassium hydrate is much more active, and this appears to me to be an added argument in favor of the use of potassium cyanide in the treatment of silver ores, instead of the sodium salt, since the presence of other alkaline hydrates, such as calcium hydrate, would in the natural course of the process, liberate corresponding quantities of potassium hydrate. In fact, the activity of potassium hydrate explains perfectly the beneficial effect derived from the use of the leach liquor from wood ashes. During the year 1905, at which time I was in charge of the treatment at the Creston Colorada mill, at Minas Prietas, Sonora, I began the use of wood-ash leach, through the advice of M. F. Perry, manager of the Grand Central at that place. He used it simply as a substitute for lime, which is generally of poor quality in Mexico. The manner in which I began the use of the leach was to introduce it into the crushing solution. We were much surprised at the resulting highly increased activity of the crushing solution. I could not account for it on any known chemical grounds at the time, but believed it due to the action of the potassium carbonate upon the large excess of zinc salts present. The above experiment, however, thoroughly explains why some operators find a preliminary treatment of lime beneficial.

To return to the question of economical treatment, I found in the Guanacevi district many rebellious ores which would yield only from 40 to 70% of their silver to plain treatment, and this recovery was not increased even when agitation was carried on indefinitely. The work in question was being done for E. F. Knotts, proprietor of the Anita custom mill, and the successful handling of this class of material meant a profitable increase to his tonnage, apart from the advantage to local mine operators. The process formerly in use at this mill was chloridizing roasting with pan-amalgamation. The idea of using pan-amalgamation in connection with the cyanide process naturally suggested itself, and it was simply a question of how the two could best be combined. It is well known that considerable gold is volatilized in giving an ore a chloridizing roast. Most of the refractory ores in question carried sufficient gold to make such a loss a serious one. Tests were therefore made on the following

lines and the resulting extractions proved comercially available: (1) crushing in solution; (2) ordinary cyanide treatment; (3) chloridizing roasting of the residues, with pan-amalgamation.

While an ore must contain sufficient gold or silver to stand the heavier costs of such a combined process, the resulting recovery is surprising and the added cost for the chloridizing roast is not as prohibitive as one might at first believe. The residue discharged from the vacuum-filter should be allowed to dry, as far as is practicable, in the sun, and such drying is quite thorough within a reasonable length of time in the case of a totally slimed ore, containing a large proportion of finely divided crystalline material. Some moisture is beneficial in the subsequent roasting operations, and the net result is the recovery of most of the gold and practically all of the silver. This method answers well for any ore carrying antimony or arsenic, and its availibility is simply a question of what sort of treatment cost the ore will stand. It will of course be understood that I am not speaking of copper-bearing ores, but of refractory silver ores, which yield only small percentages of the valuable metals to ordinary treatment. For the less refractory unoxidized ores of the camp, partial concentration, fine grinding, with 24 hours agitation, and a thorough removal of soluble values on the vacuum-filter, was found to answer perfectly.

A study of high-grade samples leads to the conclusion that the readiness with which a silver ore responds to treatment, depends almost wholly upon the mineralogical form in which the silver is present, and, apart from the question of silver locked in particles of galena, the presence or absence of other mineral constituents in the gangue has little or no effect. In confirmation of this view it will be noted that in one and the same sample of ore the gold yielded easily, while the silver proved refractory, and this was true in many different instances.

NICARAGUA AND ITS GOLD INDUSTRY

By T. Lane Carter

(August 12, 1911)

The metallurgy of gold in Nicaragua has been decidedly unsatisfactory. It is not a 'free-milling' country, and for years on some properties the extraction of the gold by plate amalgamation was from 33 to 45%. The introduction of the cyanide process was a great advance, but, unfortunately, only the sand has been successfully treated and not the slime. At one property, the Bonanza mine, the slime is treated by an entirely unsatisfactory method, the decantation process. It is questionable whether this process should be used anywhere. It is certainly a most unsightly misfit in Nicaragua.

While the slime problem presents same difficulties, on account of the large amount of alumnia present, I am sure that the slime problem in Nicaragua can be solved. Laboratory experiments gave

me an extraction as high as 90%. The only serious trouble was in clarifying the gold-bearing solution. The value of the slime is about equal to that of the sand, running about $2 per ton, and amounting in quantity to 50% of the tonnage crushed.

On the whole, it will be found that such machinery as the Huntington mills will be more suitable at the present time in eastern Nicaragua than gravity stamps. Where these muddy ores are crushed with stamps the efficiency is low, as the material sticks to the shoes and dies, increasing the weight of the stamps, simply augmenting the trouble. To my mind the best method of crushing the typical Nicaraguan ore is to pass it through crushers, then rolls, and then through the Huntington mills. As the ore is rather porous, there is no necessity for fine grinding, so that tube-mills will not be required in Nicaragua. I have obtained an excellent extraction by crushing with a 20-mesh screen.

The scheme of crushing in a cyanide solution and doing away with all amalgamation should be carefully considered, especially in the Pis Pis district, where the ores are amenable to the cyanide process. The fact that the amount of gold recovered in the Pis Pis district never equalled more than 50% and is often as low as 33% of the gold in the ore, goes to show the improtance of direct cyanidation, in the treatment of these ores. The consumption of cyanide varies from ½ to 1 lb. when treating the ores in the oxidized zone. Occasionally bunches of sulphides of lead and zinc are found, and if these pieces get into the mill the consumption of cyanide is increased. As these sulphides are not, as a rule, high in gold, it is profitable to sort these out before the ore goes to the mill.

Ordinary cone separators are used to divide the sand from the slime. It is of great importance to make a complete separation if a satisfactory extraction of the gold is desired. In one case that came to my knowledge, the people ran the whole product from the mill into the sand tanks. The amount of gold obtained from treating the mill pulp by this method was, of course, insignificant. A man who knew his business came along, made a careful separation of the sand from the slime, and had no trouble in obtaining an extraction of 88% of the sand.

The question of treatment tanks is a very important one. As a rule, they are built of wood imported from Louisiana or California. At the La Luz mine they have erected steel tanks at a great expense. Provided the property will last for a number of years, steel has the advantage of durability. With such splendid woods as are obtained in Nicaragua, one would think that the tanks would have been constructed from the native wood. This has not been done in the past on account of the absence of sawmills, but in the future, especially in new districts, it is probable that the tanks will be built from the wood of the country.

On account of the high cost of transportation, it is impracticable to use lime as a neutralizer. The vast coral reefs along the coast will give a cheap and abundant supply of lime in the future when railroads are constructed. At present caustic soda is used in the

cyanide plants. The strength of the solution varies from 0.2 to 0.3%. Occasionally a little trouble is experienced with arsenic, but T. W. Bouchelle, the chemist at the Lone Star mine, overcame this difficulty, when he noticed a deposition of arsenic on the zinc shaving, by heating up to solution in the boxes with live steam. The heat caused the arsenic to scale off the zinc and settle at the bottom of the boxes. Now and then bunches of copper are found, and these are likely to cause trouble if the cyanider does not understand how to treat the problem. By the use of bichloride of mercury in the boxes the trouble from copper can be averted.

Sulphuric acid is used in the clean-up. The transportation of acid in a tropical country like Nicaragua is a difficult and expensive feat. The gold-bearing material from the clean-up, however, is so base, that to get the gold out without the use of sulphuric acid is a serious problem. The assay offices of Nicaragua are equipped to take care of the routine work. The gasoline furnaces, such as the Braun, have been a boon to the country, and it is possible to make a gold assay in these furnaces in the wilderness for 10 cents an assay and less. On account of the heavy rainfall, which varies from 125 in. per annum in the Pis Pis district to 300 in. at Greytown on the coast, the damage done to balances and survey instruments by excessive moisture is considerable. Assay textbooks, etc., rapidly deteriorate through mildew and must be kept wrapped in cloth if they are to be preserved for a number of years.

COLORADO METALLURGICAL PROGRESS

By P. H. Argall

(January 6, 1912)

In reviewing the metallurgical processes in operation in Colorado at the close of the year 1909, I emphasized the following changes then taking place: (1) The decline of the smelting industry, due to the scarcity of high-grade ore; (2) the decadence of the chlorination process as a factor in the treatment of Cripple Creek ores; (3) the rejuvenation of the cyanidation process. At the close of the year 1911 the corresponding situation is: (1) The smelting industry is about holding its own at the 1909 tonnage; (2) the chlorination process is no longer a factor in the treatment of Cripple Creek ores, the Standard mill, handling about 8000 tons per month, having closed December 1; (3) the cyanidation process now occupies first place, with the Golden Cycle and Portland mills at Colorado City and Stratton's Independence and the new Portland mills at Victor. The Golden Cycle has treated an average of 25,000 tons per month throughout the year, the Portland 6000 or more, the Stratton's Independence 8000 or 10,000, and the new Portland 6000 to 10,000 tons. Both the Portland and the Standard plants have been re-working the tailing from old chlorination mills by the cyanidation process at substantial profits, and the Portland is now eliminating chlorination and changing the entire mill to cyanidation.

MILLS FOR TREATING CRIPPLE CREEK ORE

Name.	Tons capacity.	Process.	Remarks.
Lawrence	100	Chlorination	Destroyed by fire
Gillette	100	"	" " "
Economic	300	"	" " "
El Paso	150	"	" " "
National	100	"	Dismantled
Philadelphia	250	"	Obsolete; closed
Union	300	"	Closed
Standard	400	"	About 60% capacity
Portland	300	Roast-cyanide	To capacity
Golden Cycle	1000	" "	To capacity
Metallic	350	" "	Dismantled
Dorcas	100	" "	Destroyed by fire
Brodie	120	" "	" " "
Isabella	100	Cyanidation	Closed
Wild Horse	100	"	About capacity
Anaconda	100	"	About half capacity
Independence	350	Concentrating-cyanide	To capacity
New Portland	500	" "	" "
Jo Dandy	100	Cyanidation	" "
Ajax	200	Clancy process	Starting
Trilby	100	Roast-cyanide	Closed
Little Giant	50	Cyanidation	Starting
Homestake	50	"	"

The past two years have been active ones in the metallurgy of Cripple Creek ores. The valley plants still treat 80% of the ores produced, though division according to processes shows a change. Of the ore sent to the valley plants in 1909, about 50% was treated by chlorination; in 1911 fully 70% has been cyanided at the Golden Cycle mill. The small cyanidation plants in Cripple Creek district proper, those designed to treat oxidized ores, have not accomplished much. The Wild Horse was the only one in fairly continuous operation throughout the two years. The Wishbone roast-cyanide mill closed down early in 1910 and was destroyed by fire August 13, 1910. The Trilby roast-cyanide plant mill has not been operated at all during the period, and the Anaconda has been but lately re-started. The Kavanaugh mill on the Jo Dandy, and the Ajax Clancy-process mill are new.

A big feature of the two years in the Cripple Creek district has been the increase in milling of low-grade ore. Each year the district is producing a larger tonnage of lower-grade ore, and treatment methods must be devised to meet this condition. The valley plants, wth a dollar a ton freight for a 30-mile down-hill haul, have about reached their limit in the matter of freight and treatment charges, as a comparison of the following tables will show:

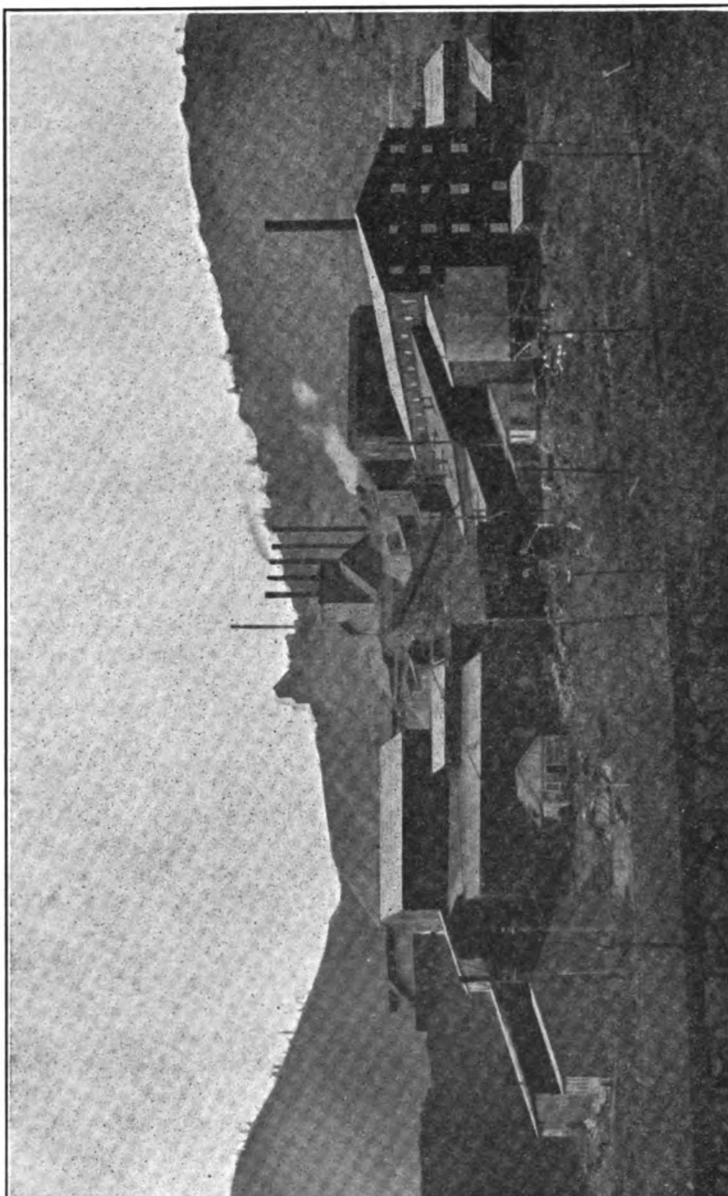
FREIGHT AND TREATMENT

1909 Oz. Au	1909 Per ton	1911 Oz. Au	1911 Per ton
Up to ½	$4.00	Up to ½	$4.00
½ to ¾	5.25	½ to ¾	4.50
¾ to 1	6.00	¾ to 1	5.00
1 to 1¼	6.50	1 to 1¼	5.50
1¼ to 1½	7.00	1¼ to 1½	6.00
1½ to 2	7.50	1½ to 2	6.50
2 to 3	8.50	2 to 3	7.00
3 to 5	8.75	3 to 5	8.00

That this scaled rate, at best, no more than covers expenses on the lower grades is borne out by the following significant statement from the last annual report of the United States Refining & Reduction Co., operating the Standard mill: "An increase in tonnage was not possible, nor was the management able to obtain treatment charges giving any material profit over and above actual expenses."

Of the mills in the district, the Jo Dandy, Wild Horse, and Anaconda are simple cyanidation plants, and having oxidized ores to deal with, present nothing new. Stratton's Independence and the new Portland, being designed to treat low-grade-sulpho-tellurides, mark the new metallurgical strides, hence more or less detailed accounts of these two mills will be presented.

Stratton's Independence.—The process, or combination of processes, elaborated by Philip Argall in the spring of 1907, and laid before and approved by the directors of Stratton's Independence, Ltd., in June of that year, provided for:



STRATTON'S INDEPENDENCE MILL, WITH AJAX IN BACKGROUND

1. Crushing the ore in cyanide solution instead of water in order that the cyanide should begin dissolving gold from the moment that fine crushing began.

2. Removing the rebellious tellurides as completely as possible by a careful concentration conducted alike on sand and slime.

3. Leaching the sand in ordinary tanks to effect a further extraction and to wash out the remaining traces of cyanidation.

4. Treating the slime by air-agitation and bromo-cyanide or other oxidizers as and when required. (A long and thorough series of full working tests has since established that bromo-cyanide is the best solvent, though even bromo-cyanide is at times useless, and at all times requires the most careful chemical application.)

In Mr. Argall's report of April 19, 1907, made after the completion of his tests on the dump ore, he showed that an extraction of 70% could be obtained at a total cost of $1.52 per ton treated, on a basis of milling 10,000 tons per month. This is not a high recovery, but anyone who has struggled with Cripple Creek sulpho-telluride ore will admit it is quite a feat to obtain 70% from $2.75 unroasted ore, yet, as a matter of fact, actual milling results have bettered the estimates made before the plant was designed, full details of which are given in *The Mining Magazine* for November, 1911.

New Portland.—The second important mill of the district, the new Portland, began operations in June, 1910, with a secret process, of which details have not yet been made public. After various changes during a 'trying-out' period, the mill settled down to steady work. The process now used includes crushing and concentrating in cyanide solution, discarding the sand after concentration and water-washing, air-agitation of the slime aided by a halogen cyanide when necessary. The concentrate produced in this mill is shipped to the Colorado City plant of the Portland company for roasting and cyanidation; the cyanide precipitate is also worked up into bars at the same plant. The costs and the recovery are closely guarded; in the annual report of the company receipts at the mine and the two mills are lumped together in one item, "cash receipts from operation of mine and mills." The local papers, however, under date of October 19, 1911, give the following: "In the last three months, ending September 30, 29,599 tons were treated, resulting in a profit of $33,110.78, averaging better than a dollar a ton for the owners." The Portland is now a splendid mill, well managed, successfully operated, and will proceed in time to greater successes, but not greater than the expenditure deserves, or than all metallurgists wish the able and energetic staff.

Ajax.—The big experiment of the year, the Ajax Clancy-process mill, started early in November and is still undergoing alterations and adjustments. The Clancy process is a radical departure from standard practice on Cripple Creek low-grade ores, in that concentration is to be eliminated and the recovery to be made by straight chemical methods aided by electricity. This is the most recent Clancy process, and its

*Abandoned, May, 1913.

merits or demerits remain to be seen. Mechanically the mill is rather complicated, which is to be regretted, since a new process is to be tried, but this apparently does not dampen the enthusiasm of A. W. Warwick, the chief metallurgist, who has already predicted the chemical consumption at 12c. per ton and the total cost per ton at $1.20. The formal beginning of continuous milling operations at the Ajax mill is awaited with expectant interest. Everyone in the district wishes a success may be scored by the new process, fully up to the claims made by Mr. Warwick, as in that event a notable advance will be marked in Cripple Creek metallurgy. None of the methods now in use is perfect, and the profession will welcome any improvement to increase the profit in milling low-grade ore. One of the cyanidation-concentration mills producing a high-grade smelter product has got the cost of treating the concentrate down to 10c. per ton milled; this, plus the small cost of concentrating, is an expense the Clancy process seeks to avoid, and to do so it may be necessary to crush the ore to pass 150 or possibly 200-mesh screen aperture. Hence, unless much higher extractions are obtained, the gain is not worth the candle.

Isabella.—An attempt was made during the year to work over tailing accumulation of the Isabella mill, but without success, the plant now being idle.

Summary.—Taking the district as a whole, the following general summary may be made. There is an ever increasing quantity of low-grade ore being produced which will lead to the building of many more mills in the district. From present indications the method in use at the Independence mill is the most likely to be followed, due to its cheapness and ease of application. The valley mills will soon suffer a decrease in ore-supply due to home milling, and to the fact that concentrate, to be produced in the low-grade mills, is a smelter product. A further point to be noted in the cyanidation mills is that the mechanical means employed have about reached their highest stage of usefulness, and that any further progress must be along chemical lines.

CYANIDATION AT TREADWELL

(June 29, 1912)

Gold ores at the Treadwell mines yield about half their value by amalgamation. In the case of the Alaska Mexican, the average percentage of recovery as free gold has been 46.66. In the mill of the 700 Ft. Claim the percentage has been 52.95. From the plates the pulp goes to vanners and the remaining saving is in the form of a gold-bearing pyritic concentrate. Formerly this was shipped to Tacoma for smelting, but now it is treated by cyanidation in a plant on Douglas Island, owned jointly by the Alaska Treadwell, Alaska Mexican, and Alaska United companies in the proportions of 60, 20, and 20%, respectively. The plant was described in 1911* by W. P. Lass, the capable superintendent, and the description need not be repeated. In

**Mining and Scientific Press,* October 21, 1911. (Reprinted, p. 272.)

OPERATING COST PER TON, TREADWELL MINES, MAY 16 TO DECEMBER 31, 1911

Month Ending.	*Total Tons Treated.	Average Value of Concentrates Received per Ton.	Average Value of Tailings Discharged per Ton.	Estimated Extraction Per Cent.	Labour.	Electric Power.		Pebbles.		Cyanide.		Lime.		Zinc Dust.		Sundry Supplies.	Total Cost per Ton.
						K. W. Hrs.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.	Pounds.	Value.		
1911.		$	$		$		$		$		$		$		$	$	$
June 15th	2,113·00	52·4189	1·7634	96·6	1·2213	26·69	·1216	15·08	·1900	2·46	·5168	7·19	·0647	2·05	·1586	·4838	2·7068
July 15th	1,600·00	59·4549	2·0519	96·6	1·5609	26·47	·1100	14·62	·2018	3·08	·6765	8·50	·0801	2·63	·2245	·5255	3·3793
Aug. 15th	2,280·00	68·1625	2·0276	97·0	1·1810	23·66	·0954	19·59	·2859	2·63	·5526	6·49	·0594	1·84	·1427	·3545	2·6224
Sept. 15th	2,316·99	63·2121	2·2503	96·4	1·1138	26·56	·0495	19·70	·2789	2·50	·5257	6·56	·0593	3·51	·2888	·4007	2·7167
Sept. 30th	1,330·00	57·2452	2·3467	95·9	0·9389	27·46	·0601	27·35	·3972	1·37	·2842	7·82	·0696	3·10	·2412	·3542	2·3454
Oct. 31st	2,641·00	61·6594	2·0150	96·7	0·9772	33·00	·0950	26·96	·3724	1·93	·3827	8·03	·0715	3·78	·3111	·6009	2·8108
Nov. 30th	2,558·59	59·2367	2·3805	96·0	1·1452	31·72	·1730	22·37	·3824	1·95	·3696	7·81	·0691	3·25	·2067	·6306	2·9756
Dec. 31st	2,911·52	60·4150	2·1841	96·5	1·0907	26·68	·1276	17·52	·3339	2·23	·4812	7·00	·0657	2·20	·1552	·6241	2·8684
Total and Averages	17,751·10	60·4375	2·1123	96·5	1·1349	27·98	·1076	20·36	·3090	2·28	·4725	7·37	·0669	2·79	·2147	·5030	2·8115

* This column is the number of tons ground and is not the same as tons received or tons discharged.

the reports of the Alaska Mexican and Alaska United companies now available, the following interesting details regarding operation for the first half-year are made available:

OPERATING COSTS PER TON, MAY 16 TO DECEMBER 31, 1911

Operation.	Labour.	Power, Air and Steam.	Supplies.			Sundry Supplies.	Totals.
			Item.	Amount.	Cost.		
	$	$		Lbs.	$	$	$
Grinding	0·1123	0·1268	Pebbles ...	20·36	0·3090	0·0207	0·5688
Cyaniding	0·1955	0·0377	Cyanide ...	2·28	0·4725	0·0044	0·7770
			Lime... ...	7·87	0·0669		
Filtering	0·1756	0·0818	Kelly Bags ...	—	0·0076	0·0106	0·2350
Precipitation	0·1198	0·0217	Cloth ...	—	0·0208	0·0145	0·4846
			Lead Acetate	—	0·0486		
			Zinc Dust ...	2·79	0·2147		
Refining	0·2164	0·0219	Sulphuric Acid	—	0·0564	0·0481	0·3778
			Fluxes ...	—	0·0850		
Superintendence ...	0·1284	—	—	—	—	—	0·1284
Repairs	0·1166	—	—	—	—	0·0334	0·1500
Assays and Accounting	0·0758	—	Assays ...	—	0·0478	0·0313	0·1549
TOTALS... ...	$1·1849	$0·2394	—	—	$1·2748	$0·1629	$2·8115

METALLURGY AND THE RAND

(Editorial, June 8, 1912)

American metallurgists are highly complimented in an article on 'Metallurgical Progress on the Rand' appearing in *The South African Mining Journal,* April 13. That there has been great improvement in Rand practice is evidenced by the decrease in cost and increase in recovery from the 85 to 86 per cent of pre-war days, to the 95 to 96 per cent obtaining at present. Despite this, our contemporary points out that technical progress on the Rand has been due more to adoption of devices invented elsewhere, particularly in America, than to local invention. The work of Mr. W. A. Caldecott is cited as exceptional, and indeed his contributions to technology have deserved and received wide recognition. In the main, it is insisted, recent improvements in cyanide practice have come from America. Pachuca agitators, Merrill filter-presses, and Merrill zinc-dust precipitators, Treat agitators, Dorr slime thickeners, are all quoted in evidence. The work at the Portland mill at Colorado Springs, including the building of large revolving-drum filters, giving better washing and deeper submergence of solution, is an instance of the activity of American metallurgists in improvement of cyanidation. Such recognition coming from a foreign technical journal of high standing is flattering. To the list cited by the *Journal* many names might be added, for the metallurgists of North America, including the United States and Mexico, have, in fact, made important contributions to cyanide practice. They have made the process practical under widely different conditions, and, what is more to the point, they continue to devise and invent.

If American engineers have been more inventive than those of South Africa, something must be credited to the genius of the people, and something to differences in conditions. At first glance

it would seem that South Africa and North America offer the same inducements to invention. Both are still in the stage of pioneering, both have been settled by mixed races. New conditions are met in each land by men of many types and of widely varying experiences. In the long run the Union of South Africa and the United States of America seem sure to develop much in common. At present they are both great mining countries. When the possibilities of South Africa come to be recognized and the attitude of the people toward colonists changes slightly, reclamation will make possible there, as in the Western United States, a large agricultural development. At present the burghers look upon the division and sale of land as almost treasonable, and in the absence of a working population of small landowners, physical labor by white men around the mines is discouraged. Men become inventors because of the wish to minimize their work, and where it is possible to "let George do it" there has not been the acute personal incentive to perfect a machine. There being no necessity, few inventions have been born.

Other important factors, relating to gold-mining especially, have been the organization of the industry and the simplicity and great size of the Rand orebodies. It was early seen that the field was unique in the security offered as to amount and uniformity of ore reserves. Any standard practice was profitable, and methods soon reached a degree of perfection that assured a good profit. The rush was then toward large-scale operations. On this the only check has proved to be the limited supply of native labor. In the development of the Rand the London share market has been the controlling motive. Anything which threatened the immediate price of shares was not to be thought of, however much it might promise for the future. The great scale of operations made and change in treatment expensive, and, if the new plan proved unsuccessful, thousands of dollars would be lost before the result could be determined. In the meantime shares would have been unsettled; and that fact was kept always before the Rand manager. These difficulties are successfully met elsewhere and might, we believe, have been better met on the Rand. At the Alaska Treadwell, after the cyanidation plant was completed, a question arose as to whether it would not be better to abandon amalgamation and send all material directly over vanners to the cyanide plant. To test the matter, 20 stamps were at once set aside and the flow was rearranged on the new plan. These stamps have now been fed for nearly a year with the same ore as has gone to the 20 next them operating according to the ordinary plan. In a short time it will be possible to tell exactly whether the new or old plan is better, and in the meantime routine operations have not been interrupted. While tests on a large scale have been run on the Rand, not enough of such work has been done, and in general it has been merely the trying out of practice developed elsewhere. Even in this there has been surprising conservatism. Butters filters, despite Mr. Charles Butters' intimate connection with Rand affairs, have only just been introduced, and only now, under the guidance of Mr. F. L.

Bosqui, is zinc dust precipitation coming into its own in South Africa. The margin of possible additional saving on the Rand is admittedly small, and the necessary capital expenditure considerable, but the tonnage still to be treated is enormous, and nothing less than the best should be tolerated. Cyanidation was first successful on a large scale on the Rand, and we would not rate the contributions to metallurgical progress made by Rand metallurgists as low as does our South African contemporary, but still it has been less than is in consonance with the magnificent scale of things in the world's greatest goldfield, and there has been far too much of truth in current gibes at the 'crusted conservatism' of the Rand. The field is no longer young; a generation, as professional practice goes, has been trained in South Africa; practice in South Africa has borrowed heavily from all the world, and payment should be more in kind and not alone in coin.

METALLURGY AT BENDIGO

By M. W. von Bernewitz

(August 17, 1912)

Bendigo, the famous old mining district of Victoria, Australia, has had a bad name in the past for its metallurgical methods, yet any careful student is forced to admit that although plants have been and are out of date, and costs of milling are high, the results are astonishing. I recently spent three strenuous days inspecting mills and discussing treatment with local engineers. I was rewarded. About $360,000,000 of gold has been produced from the Bendigo field, and it has not been found payable to treat the tailing dumps. It is quite safe to say that the ore from the Bendigo mines will never require any different process from that at present in vogue, yet the plants at work need much alteration to effect cheaper results. The field may be said to be in a transition stage from the old to the modern type of mill; and it is pleasing to note such a movement, especially now that the field has shown good returns on recent exploratory work in the upper levels and sidelines of reef. However much the Bendigo mills are obsolete, they are well kept and clean. The total cost of treatment in the old mills is about 80c. per ton.

It is probably the simplest ore in the world to treat; and is practically similar from each line of lode down to 4000-ft. depth. It mainly consists of a clean white quartz in which the gold occurs in a somewhat coarse form, while it also contains about 2% of arsenical pyrite, which produces some 5% of the total gold output. Nothing but simple treatment is necessary, since amalgamation in boxes and on plates, with a table of some kind, will extract practically everything. As an example of this, one mill may be cited which is crushing ore worth $12 per ton, and the residue after this treatment is only worth 50c. The residue from other plants runs from 16 to 50c. per ton. In the past, the saving of the pyrite was not as good as it is now, and a characteristic feature

of each surface plant is a shed containing canvas tables worked by Chinamen. These metallurgists did fairly well for a time; but not so well now. Their methods will be described later.

About two years ago a syndicate sent engineers to Bendigo with the object of sampling the old tailing heaps, which are as prominent on the landscape as they are at Kalgoorlie. If worth enough, it was proposed to erect a large central plant, transport the tailing by some cheap method, and cyanide it. The quantity was there, but not the quality. Cost of transportation would have been fairly high, and the business was dropped. A cyanide plant was erected some time ago, but could not be made to pay. One trouble was the acidity of the sand, due to decomposition of pyrite, and the consequent necessity of using much lime.

From the mines to the mills, the ore is carried by rope-ways, carts, and trucks. The use of rock-crushers is extending, and many mills are provided with a gyratory of about the No. 3 Gates size. At the majority of mills feeding is still done by hand, one man to 10 or 15 stamps. This is one of the costly features of a Bendigo mill. I have heard it stated that a good hand feeder is equal to an automatic machine. To some extent, on certain ores, this may be so; but only under very exceptional conditions. Challenge feeders are used at Bendigo, and several local machines, one of which seems to me to be fearfully clumsy. Save two, every mill visited was of the iron-frame type. In them a row of cast-iron columns or posts carries the cam-shafts and guides, and a row behind carries the driving shaft, which is on a level with the former. The driving shafts are mostly direct-connected with the engine crank and drive the cam-shaft by gear and clutches. Gear of all sizes is used, and there must be a deal of friction from this source.

The mortars of a very old design, being deep for good amalgamation and having a discharge up to 5 inches. They are built to receive 4, 5, or 6 stamps in each. Cam-shafts carry 10, 12, 15, or 24 cams; in the last case, two being coupled together. Breakage of cam-shafts is rare, as is also breakage of stems. If anybody wishes to see a variety of cams at work, let him go to Bendigo. There are the long-arm type, the separate boss and arms fitted together on the bedpost principle, half cams bolted around shaft, the Spier's dovetailed, and the modern Blanton. All cams of the old style are keyed and show much ingenuity in design, the effort being to secure ease of removal. Dovetailed cams seldom work loose.

The stamps on the field would average 850 lb. weight, with 80 drops of 8 in. per minute. They crush from 2 to 4 tons each per day. The tappets are raised or lowered on a large thread on the stem, and have renewable wearing faces. Each stamp has a separate iron guide. A regular order of drops is attempted, but in a 6-stamp mortar I noticed the two centre ones dropping together, then the outside ones, and last the second pair. Heads, shoes, and dies are made locally, and last up to six months. The speed varies at times. Punched and wire screens from 14 to 40 mesh are used.

It would seem that power on an old-type battery would be high, but one old millman and engineer assured me that his 60 stamps took only 40 hp., or 2/3 hp. each. Talking about old batteries reminds me of a fairly recent comparison at Kalgoorlie, where a worn-out battery cost $1200 in power against $500 in a modern mill on the same tonnage, which is rather a difference.

Careful attention is paid to amalgamation. A high percentage is caught in the boxes. For instance, in a lot returning 127 oz., 104 oz. was found in the box. Several short plates, in all up to 12 ft. long, with riffles between, are used. These are invariably kept locked. Berdans and retort furnaces are found in each clean-up plant, some mills having as many as five furnaces to deal with small lots from tributers or individual mines. The general amalgam retorts as high as 66%, and the average value of the gold is $19.50 per ounce.

The pyrite averages 2% of the ore, and carries 5% of its value. In the past it has been caught on the Halley percussion table, which mainly consists of a table about 5 by 3 ft., knocked forward by a three-arm cam and back by a flat spring. Its work is fair—one table for 4, 5, or 6 stamps. Most of the batteries now have installed the Phoenix-Weir table, after trials against the Wilfley, Card, and others. Its driving gear and deck are somewhat similar to the Wilfley. One table to 10 stamps is the rule. A small copper plate is fixed to the pyrite discharge of some tables, and collects a little amalgam. In some cases the overflow from the tables flows over blankets, which catch a little more mineral. All concentrate is sold to public works. Another feature of Bendigo mills which crush for the public, are the separate bins of pyrite from each lot, along with the coarse pannings from the boxes. Concentrate averages $40 per ton.

Two well known works (Leggo's and Edwards') buy concentrate from the various batteries, and I was able to see a section of each plant. The pyrite contains an average of 12% arsenic, and the method in each is to roast, following by cyanidation at Leffo's, and chlorination at the Edwards' plant. In the former works roasting is done in Leggo's patent furnace. This consists of four superimposed hearths, each in itself being a complete furnace with fire-box, feeder, and discharge to push-conveyor. The floors have an area of 60 ft. by 6 ft. 9 in. For rabbling the pyrite there are 16 spindles, worm-driven at 1 r.p.m., with 4 rabbles, one on each floor; 64 in all. The rabbles are water-cooled. The furnace capacity on concentrate is 30 tons per day, and 100 tons on an ordinary sulphide ore with, say, 4% sulphur. At the Edwards there is a simplex, and one duplex Edwards furnace, the latter 79 by 11¾ ft., traveling at about 1½ r.p.m. The hearth has a ⅝-in. fall per foot. There are about 600 ft. of chambers to catch the arsenic. The roasted pyrite is damped down, screened into vats of 4-ton capacity, fitted with tight lids and a gravel filter. Chlorine is made in an Edwards semi-rotary generator. The vats are worked in series.

At most of the Bendigo mills the pulp is lifted to ponds by pumps. A few air-lifts are in operation. The elevating of pulp at Bendigo has reached a high stage of development. The pumps are of the Cornish plunger type, with poles up to 16-in diameter. They are packed with hemp and old rubber, and have three clean water-jets to keep the sand from cutting. Self-priming apparatus is used which is simplicity itself. I was shown plungers that had been in use for years. The batteries only crush a few hundred tons per week, and the plungers last a long time, though they still require some attention.

As regards the elevation of pulp, it is interesting to note that Cornish pumps are used at Bendigo; 3-throw pumps, a few wheels, and bucket-elevators at Kalgoorlie; bucket-elevators at Broken Hill; wheels and bucket-elevators in the Waihi-Paeroa district; and wheels and centrifugal pumps on the Rand.

I should say that the Bendigo ore would yield 75% sand. At Bendigo the Chinaman has had the right, on a royalty basis, of saving what he can out of the pulp from the batteries. His skill in metallurgy is well known. I have seen it in Borneo and Bendigo and in the former northwest district of Sambas. In the sheds erected near the sand dumps are several tables with blankets or canvas. These catch the fine pyrite, and are disposed of to the pyrite works. Not much is made out of the treatment now. The Chinaman keeps the ponds in order.

When discussing why modern plants are not erected at Bendigo, the usual reply is that they would not do on this ore, where amalgamation is so important, not denying that they would be cheaper to work. A fast-running battery would throw out the gold from the mortars, and the plates would not catch it all. It is argued that one battery had to slow its stamp for this reason, yet, on the other hand, an up-to-date mill is being erected on the Catherine Reef; and a very good mill is at work on the Central Red, White & Blue mine. The residue from the latter is worth only 50c. The argument does not seem to hold, yet there are many opponents of the fast battery. It is largely, perhaps, a matter of experience in amalgamation.

The visitor to Bendigo sees three or four long, unbroken lines of brick stacks and head-gear, and scores of acres of dumps. The head-gear is generally of good design, the legs being made out of pipe, wood, or lattice. At Kalgoorlie, the head wheels are fixed toward the front of the top, and the shaft in the centre of the four legs; while at Bendigo the wheels are fixed in the centre, and the shaft is toward the centre of the two back legs; another example of each district following its own practice. All machinery in use is made in Victoria, either at Bendigo, Castelmaine, or Melbourne. There are Cornish, Lancashire, and water-tube boilers, consuming wood and coal. A great deal of slack coal is bought from the Government mine at Powlett. This costs $4.08 per ton delivered, as against $2.04 for wood. Forced draft is necessary at times. Newcastle coal is also burnt.

Name.	Transport	Feeders	Battery frame	Cam-shaft drive.	No. of stamps	Weight, lb.	No. of boxes	Stamps in box	Speed	Drop, in.	Tables
Virginia	trucks	self	iron	gear and clutch	50	800	10	5	80	8	5P-W*
South New Moon	trucks	Challenge	iron	gear and clutch	40	800	10	5	80	8	P-W
New Moon	ropeway, trucks, and carts	Challenge and hand	iron	gear and clutch	71	750 to 850	7 9 4	5 4 6	80	8	9P-W, 4H†
(1) Catherine Reef	trucks and carts	hand	iron	gear and clutch	64	800	8	5	80	8	6P-W
(2) Catherine Reef	trucks and belt	Challenge	wood	belt	30	1250	6 9	5 4	105	7	6P-W
Clarke's Public Mill	carts	hand	iron	gear and clutch	60	800	4	6	80	9	13H
Johnston's Reef	carts	hand	iron	gear and clutch	40	900	8	5	80	8	8H
Johnston's No. 3	carts	hand	iron	gear and clutch	20	900	4	5	90	8	3P-W
Central Blue	aerial tram	Challenge	wood	belt	20	1250	4 6	5 5	104	6	4P-W
Lansell's Fortuna	carts	hand	iron	gear and clutch	48	800	3	6	80	9	5H, 8P-W
Lansell's Bendigo	carts and trucks	hand	iron	gear and clutch	105	1000	11	5	80	8	

*P-W, for Phoenix-Weir table. †H, for Halley table.

DETAILS OF SOME BENDIGO STAMP-MILLS.

Hoists are of general good design, being mostly first motion, and some being fitted with Corliss gear. The battery engines are hardly of a highly economical type, although condensers are attached. The suction gas-engine is being introduced slowly, and two 120-hp. Crossleys are at work on the Central Blue, while others are on the Prince of Wales and Nell Gwynne, reducing costs. Engineers are paid $2.16; feeders, $1.60; and mill hands $1.92 per day.

In the preceding table are summarized the main features of the leading Bendigo mills.

In the Reserve, in the Mall of Bendigo, a bronze statue of the late George Lansell has been erected, and it stands there with a lump of white quartz in its hand, he having been called the 'Quartz King.' The mines controlled by the family produce some $30,000 per month. Lansell's Bendigo battery of 105 stamps is the largest in the field, and is a custom mill. It is also an old-timer, having been built in 1888. A remarkable feature of the plant is the huge main shaft, level with the cam-shafts, and driving the latter by gear and clutches. It is tapered slightly from the engine end to the far stamps. The New Moon has a 200-hp. Bellis & Morcom set, driving 150-kw. Westinghouse generator. This is for the mine hoists and crushers. There is also a belt-driven generator for two 10-hp. motors, driving pumps, etc. In this mill, after long tests, the Phoenix-Weir table was judged to be the most suitable. The Catherine Reef is about to start a fine new plant, which may be compared to the old one in the table given. As good a saving with the high-speed battery is expected as is now made in the old slow one.

Clarke's public mill, a clean and tidy plant, was quite interesting, and the owner was courtesy itself to me, as in fact was everybody I met on my rounds. At this mill were found hand-feeding; mortars with 4 and 6 stamps; 12 cams on a cam-shaft, and one with 24, it being made up of two 12-cam shafts coupled; about five styles of cams; 4-in. discharge; 2¼-ton capacity per stamp; yet the huge dump of tailing worth only 40 to 48 cents per ton. I was deeply interested.

The Central Blue has as good a battery as I have seen, the only faults being that the mill is a little cramped about the crusher, the roof over the stamps is rather low, and the cam-shafts seemed to be too long between the bearings and first stamp. It is a Victorian-built mill, and a joy to watch at work. An aerial tram brings the 800-lb. capacity skips to a No. 3 gyratory crusher, and the broken ore is distributed to the bins by belt. The 20 stamps were crushing 100 tons daily through 180-screen at the time of my visit. About 8 tons of pyrite is collected weekly, and the residue is only worth about 50c. on $12 ore. The plant is driven by a 120-hp. Crossley suction gas-engine, belt driving the main drive shaft, the power used being from 60 to 80 hp. The cost per ton for power was 10c. A 120-hp. Crossley engine also drives by belt a 10-drill Ingersoll-Rand compressor. Good amalgamation is effected in this fast mill, and the total costs are 40c. per ton, which may be compared to advantage with 80c. in an old-timer.

MINES OF THE REPUBLIC DISTRICT, WASHINGTON

By SIDNEY NORMAN

(August 24, 1912)

Unless all present signs fail, the gold camp of Republic, Ferry county, Washington, which passed through a 'boom' 12 years ago and subsequent abandonment of operations, will soon become as popular as it was in the earlier days. From a pessimistic community of but a few hundred souls five years ago, Republic has grown to a prosperous thriving town of perhaps 2000 population, while new arrivals are in evidence daily, and the residential facilities of the town are taxed to their utmost.

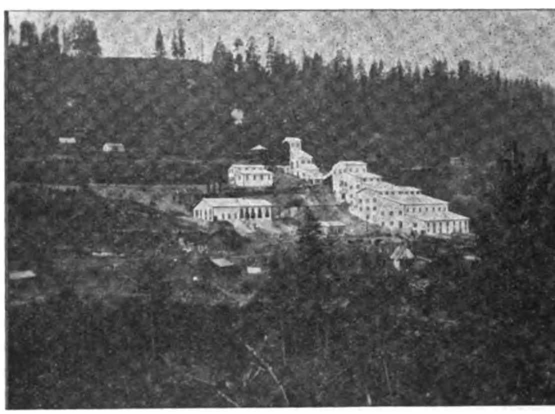
The resurrection of the camp dates from about four years ago, when J. L. Harper, who had been operating at Belcher, secured control of the old Republic group from the county, to which it had

REPUBLIC, WASHINGTON, FROM THE NORTH

reverted for unpaid taxes. Just previous to the transfer the property had been leased by the county to a partnership of miners which included James Casey, formerly superintendent under the Patrick Clark regime, and several high-grade shoots had been found and ore shipped from the old workings. Within a short time after Mr. Harper assumed control, a different aspect had been placed upon the property, and dividends amounting to $35,000 were subsequently paid. From that time forward public interest in the district revived, though the residents of Spokane, who had financed a large majority of operations in the early days, still looked with skepticism upon the movement. Money supplied by the farmers of the Palouse country has been largely responsible for the revival, and particular credit is due to T. A. White, of Colfax, who helped to finance the early operations of Mr. Harper.

A little more than two years ago dissension arose in the ranks of the stockholders of the New Republic Mining Co., which had been organized by Mr. Harper, the net result being his retirement and the subsequent haphazard development of the ground. Later it passed to the Rathfone Reduction Co., which erected a cyanide plant for the treatment of the old tailing pile, and met with some success. The property is not now being worked, but it is understood that engineers have just concluded an exhaustive examination of the old workings with the idea of consolidating the New Republic, Rathfone Reduction, and Princes Republic companies. Should such plan be consummated, the Rathfone plant will be brought up to greater capacity by the addition of adequate crushing machinery, thus giving to the camp three plants fully equipped for the reduction of its ores.

After severing his connection with the New Republic, Mr. Harper turned his attention to the Lone Pine, the Surprise, and



MILL OF NORTH WASHINGTON P. & R. CO. BELOW REPUBLIC

adjacent ground on the east side of Eureka gulch. Working bonds were secured, and the development of the ground undertaken. Since that time 46,734 tons have been sent forward to the Granby and Trail smelters, of an average value approximating $22, and the total value has been $1,067,603 gross. During last March 1841 tons was shipped averaging $21.26 per ton, while in April the output was 2475 tons, averaging $17.63. The first 862 tons shipped in May gave smelter returns of $17,364, or an average of $20.13 per ton. These are the latest figures available, but the probabilities are that the shipments since then have been of higher value, owing to the discovery of a large high-grade orebody near the point where the 670-ft. incline shaft passed through the vein. A winze is being sunk on this shoot and was down 75 ft. on June 10. At that time it showed a full bottom of quartz, and daily mine samples from June 7 to 11 gave a general gross average of $59 per ton. No returns from the smelter are yet available on this particular shoot, but mine assays or carload shipments have run around $35 per ton.

The extraction of this large tonnage of high-grade ore necessarily exposed larger bodies of ore that could only be economically treated by modern milling methods. Consequently Mr. Harper organized the North Washington Power & Reduction Co., with the eventual purpose of constructing a 1000-ton cyanide plant to treat ores of the Republic Corporation and custom ores. The first building was constructed to house machinery for two units of 125 tons capacity each, and the first of these units was placed in operation in May, under the management of Harry W. Newton, formerly connected with the Gold Road Mine, Mohave county, Arizona. No close estimate of savings has yet been made, but it is the general opinion of mining engineers who have visited the plant that the problem of economical extraction has been solved.

The new plant occupies a site on the west side of Granite Creek, within a quarter mile of the abandoned plant of the old Republic Gold Mining Co., erected in 1900 by D. C. Jackling. It is served by the Great Northern railroad from a spur running above the ore-bin level, and from this point the ore is trammed to crusher-bins, from which it is fed by a shaking-grizzly, with bars spaced at 1½ in., to a 20 by 10 Blake crusher. The product then goes to an 8 by 12 Dodge crusher and through 30 by 14 rolls to a belt-conveyor which carries it above the mill-bins to a Vezin sampler. From the mill-bins the product goes to a 6-ft. Trent Chilean mill, where it is ground to 16-mesh in cyanide solution. All material of 200-mesh overflows to the Trent agitator, the oversize going to a 22 by 5-ft. tube-mill. The tube-mill product is returned to an Akin classifier, where any not passing 200-mesh is again returned, the finished product flowing to the Trent agitator. In the first Trent agitator the pulp is thickened to the proper consistence for agitation and passed on to agitator No. 2. The overflow solution from No. 1 is returned continuously to the Chilean mill by a 2-in. centrifugal pump. The agitators, four in number, are connected in series for continuous agitation, the pulp passing from one to another by displacement. Thence the product flows to a 16-ft. Oliver continuous filter and the solution is pumped to clarifying tanks. The clear solution flows to the gold-tank and from there is fed to Trent precipitation tanks automatically fed with zinc-dust. The treated solution is pumped by a 6 by 8 Goulds triplex pump to the refinery above the mill.

Another reduction plant of 'resurrected' Republic is that of the Sans Poil Consolidated Mining Co., which acquired the Sans Poil and adjacent ground from Finch & Campbell about two years ago. Development carried on since that time has exposed large bodies of milling ore, while shipments aggregating 3300 tons, of a value of $18 per ton and a total value of $59,511, have meanwhile been sent forward. To treat this milling ore, and part of the product of the Knob Hill mine to the northeast, a 125-ton plant has been erected on the Sans Poil claim on the west side of Eureka gulch and about one mile to the north of the plant of the North Washington Power & Reduction Co.. The machinery was turned over

for the first time on June 8 and everything is now ready for the initial run as soon as track facilities at the ore-bin level have been provided by the Great Northern railroad. To prevent any delay in delivery of ore, the company has also undertaken the construction of a cable tramway that will haul mine-cars to the ore-bin level. The entire plant was designed by the Hammond Manufacturing Co. and has been erected under the superintendence of E. J. Morris. The main building is 90 by 100 ft., and rests on concrete foundations, as well as do all tanks and crushing machinery.

The mill design closely follows the Grass Valley type of continuous cyanide treatment, though novel crushing machinery has been introduced in the shape of a Williams hammer trommel mill, closely resembling the Quenner mill, used with such success in the Altar dry-placer fields of Sonora, Mexico, some years ago. An exact duplicate of the mill has been in use for the past year at the United

MILL OF THE SANS POIL CON. CO., FROM NO. 2 DUMP

States Government work on the Celilo canal near The Dalles, Oregon. There it is said to have been eminently satisfactory with a record of 20 tons crushed in one hour from coarse feed to ¼-mesh. A practical test, in my presence, on mine-run ore from the Sans Poil, resulted in the crushing of 1000 lb. to 3/16 inch in exactly two minutes, or at the rate of 15 tons an hour, or 120 tons in a shift of 8 hours. Feeding was done by hand, and the engine was not speeded up to capacity until the last few seconds of the run. If the mill can hold up to this test in daily work, it will provide sufficient feed for a 24-hr. run in one shift of 8 hours. Such a result would completely solve the problem of economical coarse-crushing; particularly when it is remembered that a large portion of the product is converted to a fine powder in the process. The product of the Williams mill is delivered by worm-conveyor to an elevator boot that raises it to a storage bin, an automatic sample being taken on the way. Thence the product goes to a set of fine-crushing

rolls, and from there it is passed on to a 5 by 12 Gates tube-mill, where it is ground to slime in cyanide solution. The discharge from the tube-mill is elevated by a tailing wheel to a duplex Dorr classifier, where a separation is made of coarse and fine material. The fine material passes to a 24-ft. diameter by 10-ft. deep Dorr thickener, and the coarse is returned to the tube-mill.

The thickened material from the bottom of the Dorr thickener passes by gravity through 10 cone-bottom air-lift agitators, connected in series and allowing sufficient contact with the cyanide to dissolve all soluble gold. The clear solution is returned to two battery-tanks for classifying purposes, the discharge from the agitation tanks flowing to a 16-ft. Oliver continuous filter of 125-ton daily capacity. The tailing is sluiced to the flat below the mill, and the solution delivered to four clarifying tanks for precipitation in twenty-four 18 by 24-in. iron zinc-boxes. Water supply is obtained from a well in the gulch and elevated to the mill by an electrically-driven power pump. Power is supplied by two boilers of 110 hp. each and a 200-hp. Erie automatic centre-crank engine. Agitation is maintained by a 10 by 16 steam-power compressor.

The Knob Hill company, largely financed by farmers of the Palouse country, is another concern that has met with success since the revival began. Its property consists of the Knob Hill and Mud Lake claims at the head of Eureka gulch, just northeast of the Sans Poil, and acquired from Jonathan Bourne of Portland, Oregon, on a bond of $125,000, with a 25% royalty applying on the bond. From this source the sum of $46,373 has already been credited on the bond, and further payments of $35,000 have also been made. Dividends to extent of $45,000 have been declared, and sufficient is now on hand to warrant the assumption that distributions to stockholders will be maintained. Within about twenty months the company has shipped 8594 tons to smelters with a gross value of $277,687, or an average of $32.31 per ton. The highest consignment in that period was 47 tons, sent forward in May of last year, which yielded $6058 gross, or an average of $129 per ton. In extraction of this higher-grade ore large bodies of good milling ore have been uncovered, and for the present 50 tons will be sent forth daily to be treated in the Sans Poil mill. If the latter demonstrates its efficiency the company will then undertake the erection of its own plant.

Several other properties well known in the earlier history of the camp, including the Morning Glory, Blacktail, Insurgent, Butte & Boston, Quilp, and others, are being worked in some degree, the payroll for May showing a total of 364 miners and surface men employed in the district. The Mountain Lion, Tom Thumb, and Rebate, all lying to the north of the main zone of activity, are at present idle, but in each case are rumors of impending sales that may result in resumption of operation before the summer is over.

The prime necessity of the district at this time is cheap power, and it is said that such arrangements are in a fair way to be perfected by Mr. Harper, of the Republic Mines Corporation and North

Washington Power & Reduction Co. If plans mature, a high-power line will be constructed to Danville, 28 miles east, where connection will be made with the lines of the West Kootenay Power Co., owning a hydro-electric plant of great capacity at Bonnington Falls, below Nelson, British Columbia, on the Kootenay river. Estimates of cost run from $50 to $60 per horsepower-year when such plant is in operation, and it is believed that present milling costs of around $3 will thereby be cut at least one dollar. Coal at $6.70 per ton, shipped from Crow's Nest is now used to generate power.

METALLURGY AT TONOPAH

By M. W. von Bernewitz

(December 28, 1912)

Simplicity of operation seems to be the key-note of treatment methods at this interesting mining centre, having been brought to such a point by much intelligent experimenting. To obtain an average of 93% extraction at a cost of, say, $3 from ores carrying 30 oz. of silver and a few grains of gold per ton is good work, and not long ago would have been termed impossible, especially in such a situation as Tonopah.

Tonopah ores may be described as consisting of fine granular quartz (the silica averaging perhaps 80%), without noticeable quantities of sulphides, poor in the baser metals, and containing disseminated silver minerals, and gold. The primary metallic minerals are silver sulphides, principally polybasite, stephanite, and argentite, with occasional pyrite, chalcopyrite, galena, and blende. Silver selenide also occurs. Silver chloride, bromides, and iodides occur, mainly at least, as secondary minerals. Silver also appears in the metallic state. Gold occurs in the proportion to silver of about 1 to 100 by weight, and has been seen in the free state.

In Tonopah there are five mills—the Belmont, Extension, MacNamara, Montana, and West End—while at Millers, 12 miles north, are the Belmont and Tonopah mills, ore being shipped to these at a cost of 70c. per ton. In nearly every case gyratory crushers are used for breaking ore as it comes from the mines, the procedure being to crush first in a large crusher, up to the No. 7½ type K Gates size, pass through revolving trommels, the oversize being again reduced in No. 3 size gyratories, the final product for the stamps being about 1¼ in. Sorting is done at the Belmont and MacNamara mills; at the former on a pan conveyor from which 15% is rejected; and at the latter on a 30-in. rubber belt, from which 6% is sorted out. From the crushing department, the ore is taken to mill bins by 20-in. belt conveyors, or bucket elevators, and distributed by the usual automatic devices.

Tonopah millmen have not been troubled with the heavy-stamp mania, although the ore is fairly hard, and the 320 stamps at work vary between 1100 lb. at the Montana to 1400 at the MacNamara,

the latter having probably the toughest ore in the district. Several different methods of driving stamps by motors are to be noticed, and will be separately described later. Foundations are usually of concrete, and give satisfaction. The new Belmont mill has sheet-lead under its mortars, but this is being replaced with rubber. There are no high-speed stamps, the average being perhaps 103 drops per mine with 7-in. drop. Both square-mesh and ton-cap screens are used, sizes varying from 6 to 20 mesh, and the stamp-duty is from 4 to 8 tons per day. The Challenge feeder is now a simple contrivance compared with the original, and may be described as a double monkey-wrench grab, which turns the gear and feed plate. There is no amalgamation at Tonopah, nor is it necessary on this class of ore. Crushing is done in weak and warm (from 50 to 80°F.) cyanide solutions, so the ore is in contact with solution from the stamps to filtration. This is necessary as well as the heating, which, although somewhat expensive, quickens the solution and accelerates the dissolving action. Solutions are usually heated to about 95°, and in one case 120°, by live steam introduced in the agitators.

The practice of using hot solutions is briefly as follows: At the new Belmont mill the temperature at the stamps is from 60 to 70°F., and at the Pachuca agitators exhaust steam from the mill air-compressor is fed in, increasing it from 90 to 100°. In the *Mining and Scientific Press* of January 27, 1912, A. H. Jones, metallurgist at this plant, gave some valuable data on this subject. On an ore carrying 0.05 oz. of gold and 18.2 oz. of silver per ton, 60 hours' agitation with both 60 and 90° solutions, the tailing averaged 0.0175 and 3.45, and 0.0125 and 1.90 oz. respectively. Tests on 48 and 69 hours at similar temperatures gave as marked results. Besides the effect on extraction, the hot solutions flowing through the mill kept the whole place at a good working temperature. At the Montana-Tonopah, ore is crushed in 50 to 60° solution, which is increased to 110° at the Hendryx agitators by live steam. It is found also that the heat aids settling. There is a marked decrease in extraction without hot solutions.

The MacNamara mill recently had experience with cold solutions, owing to an enforced shut-down for two days. The Trent agitators are usually kept at from 115 to 120° by live steam and it was found that it took several days to heat everything again, in the meantime the time of agitation had to be increased and extraction fell off considerably. Heat is necessary in the summer, but less steam is used. The cost is about 30c. per ton treated. The Extension ore is crushed in 80° solution, and live steam is added to the Trent agitators as soon as possible, making the temperature up to 120°. It was found that this was better than 90° and extraction has improved 1.5 to 2% during the past few months, it being 94.5% at present. About 2100 tons of solution is circulating in the mill, and it takes 7 days to heat this if it should get cold, meanwhile extraction falls off.

Cost of heating is 18c. per ton. It has been found at the Belmont mill, at Millers, that in passing through a tube-mill the tem-

perature of the solutions increases, presumably by the grinding action of the pebbles, mill liners, and ore particles. A test taken while I was there showed feed temperature at 65°, and discharge 70°. In the *Mining and Scientific Press* of February 24, 1912, Noel Cunningham, at Millers, contributed the results of some experiments, proving that laboratory work had shown greatly improved results from hot solution. At another plant treating Tonopah ore, crushing is done in 76 to 80° solution, increased to 95° in the agitators by live steam in coils. Recent tests showed a saving of 24 to 32c. at a cost of 11c. per ton. The same temperature is kept up in summer and winter. At the Mexican mill, Virginia City, solution is heated to about 96°, at a cost of 12c. per ton, results being improved by this system, the average extraction being 92 per cent.

It is to be hoped that millmen in Cobalt, Mexico, and Waihi will give their experience with heating solutions. When I was with the Waihi company in 1898, heating was tried, but I kept no data. It may be interesting to mention that, at Kalgoorlie, treating an ore containing practically no silver, some argument was raised as to the benefit of hot solutions in treatment. The heating there is not intentional, as at Tonopah, but comes about through the hot roasted ore being mixed with solution, bringing it up to nearly 200°. The discussion resolved into whether by cooling prior to mixing there would be less consumption of cyanide, and less trouble with sulphates being deposited in launders, pipes, and pans, or whether there were benefits derived from the resulting hot pulp. The Associated, Associated Northern, and Kalgurli, more particularly, found that there was no appreciable decomposition, and up to 50% of the gold was dissolved at that point; while the Great Boulder, Perseverance, and South Kalgurli preferred to cool the ore before mixing.

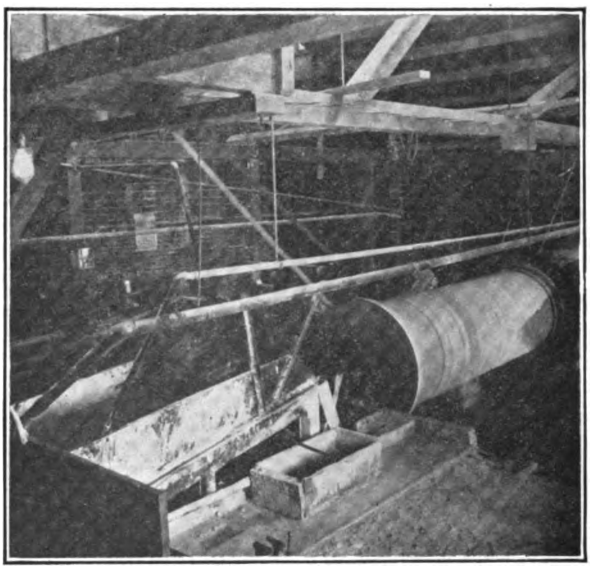
Tonopah ores carry as much as 3% of pyrite, but concentration is not always employed, it being done only at the Belmont, Montana, Tonopah, and West End. It would seem that if the grade of the ore and percentage of mineral is not too high, tables are not necessary, and this varies from time to time in the various plants. At any rate, a very close saving is not attempted. The Extension company dispensed with their Deister tables, selling them to the West End. The Belmont, Montana, and Tonopah use Wilfley tables. Concentrate is collected, steam dried in large trays, sacked, and shipped to smelters. Freight and treatment cost nearly $70 per ton. It seems a pity that such a high cost is necessary, and that the product could not be handled locally by some central plant, which would treat the combined output at about $10 per ton.

All-sliming is the standard method, with the exception of the Tonopah mill at Millers, where three products are made: concentrate, sand, and slime. At this plant reduction is by stamps, and Chilean and Huntington mills; while at Tonopah the procedure is as follows. The pulp from the stamps is fed into Dorr duplex classifiers making 12 strokes per minute, from which slime overflows and coarse material is fed into tube-mills by means of a special

feed. Discharge from these is elevated to the Dorr classifiers, and a further classification takes place, further grinding in the tube-mill, and so on. The only product which escapes to the slime plant is from the Dorr machines. This is termed the closed-circuit system, and is a good one. At the West End, the Dorr classifier discharge is:

	%
100 mesh	99
150 "	93
200 "	89

There are 16 tube-mills in the district, varying from 5 by 18 ft. to 5 by 22 ft., revolving at about 26 r.p.m. Usually the locally-

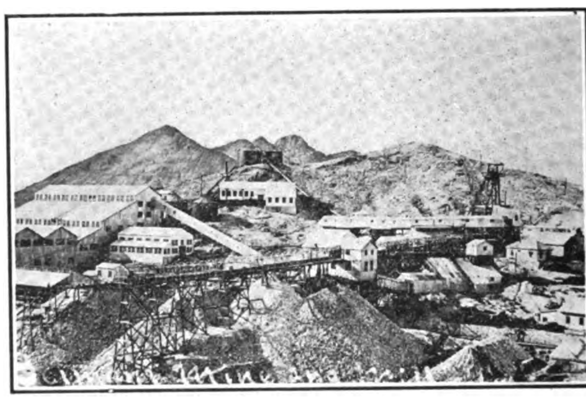


DORR CLASSIFIER, TUBE-MILL, CLOSED CIRCUIT

made smooth liners give good results. These are ½ in. thicker at the feed than discharge end, and last over nine months. At the Extension mill, there are two classifiers and two tube-mills in the closed-circuit, arranged thus in series: coarse pulp from No. 1 classifier is fed into No. 1 tube-mill which is fitted with ribbed liners, and is discharged to a bucket elevator, which in turn lifts the partly-ground material to No. 2 classifier which feeds No. 2 tube-mill fitted with smooth liners, the product being returned to the No. 2 classifier. The consumption of pebbles is 4.5 lb. per ton milled. It is claimed that ribbed liners are better on coarse pulp than smooth ones, and the latter can do the final grinding better. The ribs pick up pebbles and toss them about more in the mill, this giving extra impact for coarse material which is not necessary for

the fine. At the Goldfield Consolidated mill this is also the practice. Pebbles remain in a better rounded shape with ribbed than when used with smooth liners, which tend to produce flat surfaces. When discussing the action of various crushing machines in the *Mining and Scientific Press* of December 10, 1910, I mentioned the corrugated liners used at Kalgoorlie affording a long life and extra grinding results. Some of the Tonopah mills find that it is not necessary to get an extremely fine product, the pulp being 73 to 89% through 200 mesh.

Various types of thickeners or dewaterers are in use, the practice being to allow the clear solution to overflow and decant off as much as possible for battery storage. When it gets too high in gold content it is decanted to the tank for precipitation. As at many other mining centres there is quite a difference of opinion regarding the efficiency of agitators, the Trent, being used at the McNamara, Montana, and West End; the Hendryx at the Montana; Pachuca tanks at the new Belmont mill; and ordinary mechanical agitators and air-lifts at the Belmont and Tonopah at Millers, these being in series at the Belmont plant. Centrifugal pumps and air at about 20 lb.pressure are used for the Trent system; and better results are obtained if pulp is drawn off near the top of a full vat and pumped through the arms as usual. Agitation proceeds for upwards of 48 hours. At the new Belmont mill, slime isfirst agitated in six Pachuca tanks, and from these it is elevated to Dorr thickeners by an air-lift, prior to going to another set of six Pachucas, making a total of 48 hours' agitation, the idea being to get rid of as much valuable solution as possible before sending slime to the filter-plant. Cyanide and lead acetate are added to the agitators, the former being from 2 to 5 lb. solution, while regular addition of the acetate is found necessary at all mills. Lime is usually slacked, and added to the tube-mill feed. Consumption of chemicals at the Extension is as follows:



NEW MILL, BELMONT COMPANY

	lb.
Lead acetate, per ton	0.9
Cyanide, per ton	2.5
Lime, per ton	3.5

Agitated slime is drawn off to stock-tanks, which serve the purpose of storage from agitators and excess from filter-plants. The latter have little of special note about them, they being of the ordinary stationary leaf type which have been described so often in technical papers.

Zinc dust precipitation is used at the new Belmont and Montana mills, and zinc shavings at the Belmont, Extension, MacNamara, Tonopah, and West End. Methods of dealing with precipitate vary somewhat. At the new Belmont precipitate is dried, mixed with 5% of borax and smelted in double-compartment, oil-fired Rockwell furnaces, lined with carborundum, kaolin, and water glass. At the Extension it is dried, fluxed, and smelted in oil-fired Steele-Harvey tilting furnaces, which contain a No. 250 graphite crucible, while at the Tonopah mill the fine zinc shaving precipitate is incompletely dried, mixed with crude borax which swells up through the mass, and then is smelted in six coke-fired tilting furnaces. Crucibles last from 90 to 130 hours, and are turned once. Tonopah bullion will average 950 fine in silver, and a trifle over 10 in gold, and is sampled by being bored at opposite corners of top and bottom of bars. The bullion is shipped by freight like any other merchandise.

The mills at Tonopah are fortunate in having such a good supply of electric power as is available, and mechanical details are thus much simplified. I do not think there is a single engine in any mill there, also every machine having a motor drive. Chain drives are fast becoming popular, and apparently there is no dissatisfaction with them, once their peculiarities are understood. Such items as motor drives, and elevating pulp will be discussed later. In conclusion, it seems to me that generally the treatment of silver ores at Tonopah has been simplified to a fine point with good results, and is creditable to all concerned. There is not a dismantled mill, nor a badly designed one, nor any built for a mine without ore, to detract from the good work of the district.

ELEVATING PULP

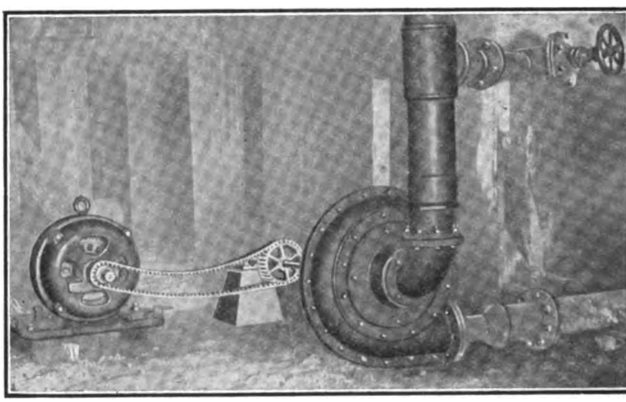
By M. W. von Bernewitz

(February 15, 1913)

In Tonopah and Goldfield milling plants quite a variety of ways of elevating and circulating pulp are employed, and a short description of these should prove interesting, as millmen are often at a loss to decide what system to use. The new Belmont mill has three-throw plunger pumps lifting thickened pulp from No. 1 Dorr tanks to the first battery of Pachuca agitators; and from these it

is elevated to the No. 2 Dorr thickeners by an air-lift. Excess from the vacuum-filter vat is returned to the stock tank by a bucket elevator with close-connected buckets. This is vertical, and part of the time is heavily loaded, but gives no trouble. At the old Belmont mill, at Millers, a tailing-wheel is used to lift pulp from the stamps a height of 48 ft. to classifiers. The up-keep of this wheel is low, and it is reliable. This plant treated 87,952 tons during the past financial year, and elevating and classifying cost about 3.6c. per ton.

From the Dorr classifier and tube-mill closed circuit in the Extension mill, slime is raised to the thickeners by a Campbell & Kelly centrifugal motor-driven pump, running at 600 r.p.m. The life of liners is from 180 to 240 days, at a cost of $35 per liner and runner. Work of this pump is most satisfactory, and it is used largely at Tonopah. It is made by the local foundry, and has attracted favorable attention. Liners are made of a special tough



CHAIN-DRIVEN PUMP

steel mixture, and are not cast split in the usual way, which permits pulp to cut them out rather fast. The accompanying illustration shows one of these pumps at the MacNamara mill, driven by a chain from motor, a style of drive very popular and satisfactory.

A 5-ft. Frenier spiral pump at the MacNamara lifts tube-mill pulp a height of about 12 ft. to a Dorr classifier. Its work is satisfactory. For circulating slime in the Trent agitators, 4-in. Campbell & Kelly pumps are used, driven by 7½-hp. motors, and consuming about 6 hp. At the filter vat there is another of these pumps, used for filling, returning excess slime to stock tank, and circulating. It is belt-driven by a 20-hp. motor and works well. At this point, all pipes are so fitted that they will drain properly after being used, as in cold weather, at this altitude, pulp would be liable to freeze. At the Montana mill Frenier pumps give good service. In the cyanide department the 6-in. Morris centrifugal pumps, lined with manganese steel, last about 18 months. They

are used for elevating all pulp to agitators from settlers, and to stock tank from agitators, and excess from filter vat.

The Tonopah mill at Millers has two wheels at work. No. 1 is 30 ft. diam. and elevates 500 tons of dry ore and 3500 tons of solution per day from the stamps. It is driven by a 7½-hp. motor. The first cost of wheels is considered high, but they are efficient, and maintenance is low compared with bucket elevators. The cost of elevating and separating is given as 8c. per ton. Campbell & Kelly pumps are used at the West End mill for circulating slime in the Trent agitators, transferring slime to the thickener, and at the vacuum-filter; and are highly praised.

The Goldfield Consolidated has two bucket-elevators lifting pulp from Chilean and tube-mills to classifiers. The elevators are each 28½ ft. from centre of tail to centre of head pulley. The housings are of steel plate, and are 28 in. wide and 76 in. long, with a total height of 32 ft. 8 in. The head pulleys are 18½ by 42 in., fitted on 3 1/16 in. by 7-ft. shafts. Spur gears have 59, and pinions 15 teeth with 2-in. pitch, and the driving pulleys are 6½ by 42 in. Tail pulleys are 18½ by 32 in., fitted on 3 7/16-in. by 4-ft. 10-in. shafts. Both head and tail pulleys are lagged with 6-ply rubber belting, secured with ⅜-in. elevator bolts, 5 in. apart. Flanges or collars on the tail pulley prevent grit getting into the bearings. The belts travel at 450 ft. per minute, and the carrying capacity is about one-third of the theoretical cubic content of the buckets. The belts are Silverton rubber make, 18 in. wide and 8 ply, and are fitted with malleable iron buckets, 8 by 8½ by 16 in., spaced 14 in. centres. Each has two ⅛-in. holes, 4 in. apart in the bottom, giving a freer discharge. Power absorbed by each elevator is 7 hp., and the life of a belt is 12 months, during which 400,000 tons of pulp is elevated, at a cost of 1.5c. per ton.

DESCRIPTION OF NOTABLE MILLS

AN EARLY CYANIDE PLANT

(January 21, 1911)

One of the first, if not actually the pioneer plant to treat ore on a commercial scale by the cyanide process, was built near the Calumet mine in Shasta county, California, in 1891, by A. B. Paul. It was thus described in the *Mining and Scientific Press,* in the issue of October 3, 1891:

"The plant is intended solely for the working of the McArthur-Forrest process, not alone on the ores of the three counties for which the company owns patent rights, but for all the ores that may be shipped from all portions of the coast. The plant will treat 10 tons of ore every 24 hours, and is so arranged that different lots of ore can be treated at the same time. In the treatment of the ore, the first operation is drying. It is then passed through a rock-breaker and into bins, from which it is fed into a Paul barrel-pulverizer, and when powdered the ore is placed in agitators and a 1% solution of cyanide of potassium added. After an agitation of 6 to 12 hours the liquor is drawn off into filtering tubs. These filters are of wash gravel covered with canvas. The liquor passes through the filter and into storage tanks. From this the solution is drawn into a chest of zinc filters, each filled with zinc shavings. The liquor flows down through the first (box), up through the second, down through the third, and so on, to the end of the eight filters. The gold is precipitated upon the zinc shavings in the form of a brown powder. When desired, the chest is unlocked and the zinc shavings washed in clear water, which separates the gold. When it has settled, the water is drawn off and the gold, in the form of brown powder, melted into bars. The liquor from the filtering tank is pumped back to the first tank and sufficient cyanide of potassium added to bring the solution up to the original 1%. As will be seen, the process is very simple, no roasting of ores is needed, and no high-priced chemicals required, with a very small loss of materials used. * * * As numerous parties have failed in making small working tests of this process, it may not be amiss to state that very often cyanide of potassium is not more than one-half to two-thirds full strength, and it is therefore necessary to know the exact percentage of cyanide as well as to follow the company's method of treatment."

This is one of the first articles descriptive of the cyanide process ever published in a technical journal. The details given are sufficiently comprehensive to make the operation of the process easily understood, as far as knowledge of it went, but subsequent development of the process proved it to be far from the simple application of a long-recognized fact—that a weak solution of potassium cyanide would dissolve gold. Since that time, the fall of 1891, tons of literature of the highest type of classical contributions to metallurgy have been published, and experimenters are still earnestly engaged in the study of methods by means of which the process

may be still further improved and its commercial application successfully extended to the treatment of those ores which are still refractory.

NEVADA HILLS MILL

By W. A. Scott

(January 20, 1912)

The new concentrating mill and cyanide plant of the Nevada Hills M. Co., at Fairview, Nevada, was put in operation September 15, 1911. It has a rated capacity of 150 tons per day and is gradually being brought up to that tonnage. There is a descent of 120 ft. from the crusher floor to the sump floor, there being eight levels, occupied as follows: (1) crusher and elevator; (2) mill-bin and stamp batteries; (3) classifiers and concentrators; (4) tube-mills; (5) collecting and thickening tanks; (6) dilution and decantation tanks; (7) treatment tanks and filters; (8) heating plant and sump. The building, set on foundation and retaining walls of reinforced concrete, is constructed of steel, and has a covering of asbestos-lined corrugated iron, making it practically fireproof. The ore is drawn from the mine-bins over grizzlies to a 12 by 20-in. Hercules crusher, by which a sufficient tonnage is crushed in 8 hours to supply the mill for 24 hours. The crushed ore is raised to the top of the mill by a 12-in. bucket elevator, and it passes thence by gravity through an automatic sampling plant and automatic weigher into a 600-ton steel-frame ore-bin immediately above the stamp batteries. It is drawn from the bin through four Hutchinson automatic feeders, by which it is fed to the four 5-stamp batteries of 1250-lb. stamps, and these are operated in two 10-stamp sections, each driven by a 35-hp. motor, belted to a cam-pulley at the centre of the section. The four mortars, weighing 12,000 lb. each are made with narrow boxes, and designed for rapid discharge. Each mortar is anchored to a concrete foundation by seven 1¾-in. bolts of Norway iron. Each battery as a unit is designed for heavy duty, the pulp being discharged through 20-mesh screens. The present method is to pass the crushed ore from the batteries to classifiers, the fine going to 14 Deister tables, and the coarse to two 18-ft. tube-mills, 5 ft. diam.; but it is now understood that a Chilean mill is to be provided for regrinding the coarse from the classifiers, and the product of the Chilean mills is to pass through an Akins classifier, the fine from which is to be concentrated over Deister slime-tables, the coarse to be reground in the tube-mills. The tube-mill product is then to be classified by an Akins machine, the slime from which is to pass to two Dorr thickening tanks, the sand to be returned to the tube-mills for further pulverization, making a closed circuit that will permit no material to pass to the thickening tanks till it has been reduced to the requisite degree of fineness. The thickening tanks, which are 34 ft. diam., 12 ft. high, are supplied with overflow launders and heavy Dorr thickeners; the thickened pulp flows to a mixing-box, where lime-water is added, and it passes thence to nine

agitating treatment tanks, each 12 ft. diam. and 32 ft. high, the clear overflow from the thickening tanks being returned by a 6 by 8-in. triplex pump to a 50,000-gal. tank near the mill. The standardized solution is put in the first treatment tank, the nine tanks being arranged in series for continuous agitation. The pulp undergoes 42 hours' treatment from the time it enters the first tank until it is discharged from the last one. A 10 by 13-in. duplex low-pressure Ingersoll-Rand air-compressor is in use for supplying air for agitation and aeration. The pulp passes from the last treatment tank through three dilution tanks, 34 ft. diam. and 12 ft. high; these are provided with overflow launders and Dorr thickeners, so arranged that the overflow from one tank passes into the next tank below it. Barren solution from the zinc presses is added in the first tank and mixed through it by the Dorr thickeners, by which the solution is diluted. The overflow, after passing to the succeeding tank, is given treatment similar to that in the first, the treatment being automatic and continuous through the three tanks; the thickened pulp is drawn off at the bottom of the last tank and transferred to the stock tank, 30 ft. diam. and 12 ft. high, equipped with a Trent agitator.

NEVADA HILLS MILL

The rich solution overflowing from the last tank is passed through two 36-in. 40-frame Perrin clarifying presses to three precipitating tanks, into which zinc dust is added by a Merrill zinc dust feeder; the solution is then delivered by two triplex pumps to two Merrill precipitating presses, each of 300 tons capacity, situated in the refinery building. The barren solution from these presses flows to two solution tanks, from which it is drawn off and standardized.

The pulp in the stock tank, above mentioned, is drawn to two Oliver continuous filters, each 11 ft. 6 in. diam. and 12-ft. face, the filtered and washed residues discharging upon a 14-in. belt tailing-stacker 200 ft. long, running at an inclination of 20°. Vacuum for the filters is maintained by two 14 by 14-in. belt-driven vacuum pumps.

The late John B. Fleming, who designed and directed the construction of the plant, made the following statement in his report to the company: "Inasmuch as concentration, when preceding cyanidation of silver ores, is being abandoned in many cases, it should be stated that there is in this mine a large tonnage of oxidized ore containing manganese dioxide so combined with silver as to make a combination that does not yield to cyanide treatment, and concentration must be adopted to remove this mineral. When the oxidized ore is exhausted concentration probably can be abandoned."

The water supplied for mill work is pumped from the mine by two Gould's triplex pumps, each having the capacity of 100 gal. per minute, and each of which is driven by a 40-hp. electric motor. The property is being operated under the management of E. A. Julian, with J. W. Hutchinson as consulting engineer.

RECENT CYANIDE PRACTICE AT THE MONTEZUMA, COSTA RICA

By S. F. Shaw

(January 28, 1911)

The Montezuma Mines of Costa Rica are situated about 15 miles from Punta Arenas, a seaport on the western coast of Costa Rica. The veins are supposed to have been worked to some extent in early days, but the earliest authentic records of work go back to only 1899 at which time two 20-stamp amalgamation mills were erected and operated. A low recovery of gold and silver was obtained, probably between 50 and 75%, as part of the gold was enclosed in the sulphides in the ore. In 1901 experiments were made with cyaniding, but along unpractical lines. In 1906 the consolidated Bella Vista and Thayer mining properties erected a 40-stamp mill which began operations in 1907 employing plate amalgamation, followed by leaching of the sands. In 1907 and 1908 earth slides and freshets carried away the slime plant then in process of erection, owing to the temporary character of the construction, which necessitated re-building. Advantage was taken at this time to introduce the latest practice in the treatment, wherein agitation in Brown tanks and filtration with Butters filters would be employed. The ore as it comes from the mine in 1-ton side-dumping cars is weighed on a 10-ton Fairbanks scale, then trammed to the mill and dumped onto a grizzly floor of 16½ by 66 ft. with bars spaced 2 in. apart. Owing to lack of head room, it became necessary to use a flat floor and to scrape the ore over the grizzly openings. The bin is 66 ft. long by 14½ ft. wide and 23 ft. deep at the lowest point; 8300 cu. ft. of space is available for the fine that has passed through the grizzly. Two 7 by 10-in. Blake crushers take the oversize, crushing same to 1 or 2 inches. Eight Challenge feeders supply the batteries with ore. The lime, amounting to 6 lb. per ton of ore, is added to the ore as it enters the feeders. The stamps weigh 1050 lb.

and drop 6 in. 96 times per minute. The screens are slotted, equivalent to 12-mesh, and last about 120 days. Double-crimped brass screen, 20-mesh, is also used at such times as the tube-mills are not in use, and last about 10 days. Crushing is done in 0.10% cyanide solution, the proportion of solution to ore being about 8 : 1. The percentage of material from the 8-mesh screens passing through various-sized screens is as follows:

Mesh.	Per cent.	Mesh.	Per cent.
20	26	100	6
30	13	120	2
40	6	150	1
60	16	200	4
80	6	—200	20

The pulp from the batteries flows through eight 3-in. pipes to two 60° conical settlers provided with Caldecott diaphragms, which de-

CYANIDATION PLANT

liver a product to the Abbé tube-mills containing 26% moisture, and sizing as follows:

Mesh.	Per cent.	Mesh.	Per cent.
40	42	120	3
60	22	150	3
80	13	200	3
100	10	—200	4

These tube-mills are 20 ft. long by 4½ ft. diam. The silex lining lasts about 10 months, and Danish pebbles are used as a grinding medium, the consumption being about 1½ lb. per ton of ore. The tube-mills are provided with a spiral feed which occasionally gives trouble by clogging, also by throwing out sand from time to time. The discharge from the mills contains 34% moisture, the amount above that in the feed being added, as pulp with only 26% moisture is too thick to discharge.

The sizing test of the discharge is as follows:

Mesh.	Per cent.	Mesh.	Per cent.
60	3	150	6
80	7	200	15
100	18	—200	46
120	5		

The discharge from the tube-mills joins the overflow from the settlers and runs two 50-in. conical settlers through two 4-in. pipes, the underflow being returned to the tube-mills by a 6-in. Price centrifugal pump driven by an 18-in. Pelton wheel operating under a head of 470 ft. A Dorr classifier was erected to take part of the flow to these 50-in. settlers with the intention of leaching part of the sands. As it was found that the slime plant could handle all the ore that was delivered to it, the Dorr classifier was not used. For treating these sands, there are 4 tanks 28 ft. 8 in. diam. by 5 ft. deep. The overflow from the two 50-in. conical settlers and the Dorr classifier flows through a 4-in. pipe to a cone-settler 25 ft. diam. and 25 ft. deep, having a slope of 60° and containing 5500 cu. ft. The upper 4 ft. of the settler is cylindrical. The sizing test of the pulp at this point is as follows:

Mesh.	Per cent.	Mesh.	Per cent.
60	1	150	4
80	7	200	10
100	12	—200	58
120	8		

The overflow from this settler goes to a tank 29 by 8 ft. containing 5550 cu. ft. This tank also receives the solution drawn from the sand-leaching tanks. When the settler is full, it is discharged into one of two Brown agitation tanks 15 ft. diam. by 45 ft. deep, having a conical bottom with a slope of 60°, and holding 6038 cu. ft. This pulp averages about 66% moisture. Agitation is given for 12 hr. in 0.10% cyanide solution and pulp is then drawn through an 8-in. pipe by a Butters 8-in. centrifugal pump and delivered to a cone-reservoir containing 5500 cu. ft., of the same dimensions as the first settler which discharges into the Brown tanks. The pulp is here kept in continuous agitation with air at 25-lb. pressure until all is drawn into the filter. Filtering is done in a 60-leaf Butters vacuum-filter contained in a steel box having hoppers with 60° bottom slopes. Three 3-in. overflow pipes carry off the surplus pulp, while a 6-in. Price centrifugal pump continuously delivers pulp to the filter during the time the cake is being formed, through a 4-in. pipe at the top and on the side of the filter-box, and distributed into the box by sixteen 1-in. pipes placed over the leaves, being spaced so as not to interfere with taking out the leaves. After a cake, averaging 1¼ in. thick, is formed, the pulp is discharged to two tanks 14½ by 6 ft. containing 990 cu. ft. each, from which it is returned to the conical reservoir; and when making a cake is also distributed over the filter, by the 6-in. Price centrifugal pump previously mentioned. The weight of dry pulp in a cake on each leaf averages 550 lb. Wash solution is admitted through the cake for

45 minutes, after which the box is filled with water and washed for 5 minutes; then the cake is discharged by allowing water under 40-ft. head to enter the interior of the leaf. The vacuum is maintained by a 14 by 14-in. Gould vacuum-pump running at 35 r.p.m. and driven by a 6-ft. Hug water-wheel operating under a head of 550 ft. The Butters pump is driven by a duplex Pelton water-wheel. The cycle of operations averages as follows:

	Min.
Filling with pulp	8
Making cake	60
Returning excess pulp	8
Filling with wash solution	8
Washing	45
Returning wash solution	8
Filling with water	7
Wash with water	5
Discharging cake	8
Total	157

MONTEZUMA MINES

The discharged cake contains about 30% moisture. Thirty tons of strong solution is drawn through the leaves per cake, and 18 tons of wash-solution, averaging 1.2 tons of wash-solution per ton of dry pulp. The filter-leaves are washed in an acid tank accommodating 3 leaves which is operated automatically, drawing the acid through the leaf and then forcing it out again, 0.2% hydrochloric acid solution being used for this purpose. The strong pregnant solution from the filter flows to a strong-solution gold-tank 39 by 7½ ft., containing 8960 cu. ft. This tank also receives the solution from the 29 by 8-ft. overflow tank. The weak solution is pumped to a weak-solution gold-tank 20 by 6 ft., containing 1885 cu. ft. The strong solution is drawn through 6 rows of zinc-boxes, 6 compartments in a row, each compartment being 24 by 24 by 24 in. and containing 5 cu. ft. of zinc-shaving. The weak solution flows through 3 rows of zinc-boxes, 5 in a row, having compartments of the same dimensions as the strong boxes. The strong solution, before passing through the zinc-boxes, averages $3.40 per ton, and after being

precipitated averages $0.49 per ton, thus securing an extraction of 86%. No effort is made to obtain a thorough precipitation. The amount of zinc consumed is 0.85 lb. per ton of ore. The strong solution flows to a sump 35 ft. 3 in. by 7½ ft. containing 7450 cu. ft., from which it is pumped by an 8 by 8-in. Gould triplex pump driven by a 7-ft. Pelton water-wheel through a 3-in. pipe-line to the battery storage-tank, 29 ft. 4 in. by 13 ft. 8 in., containing 9215 cu. ft. The weak solution is pumped from the weak-solution sump containing 380 cu. ft. to the weak-solution storage-tank, 35 by 7½ ft., containing 7215 cu. ft., by a Price 3-in. centrifugal pump.

Air is compressed by a Rand duplex 10 by 16-in. compressor, which is relieved when making repairs by a 16 by 10-in. Clayton air-compressor. The electric lighting system is supplied with current by a 75-kw., 300-amp., 250-volt General Electric dynamo, running at 550 r.p.m. The two compressors, triplex pump, and 3-in. centrifugal pump are driven by a 7-ft. Pelton water-wheel operating under a head of 550 ft. The precipitate is drawn from the zinc-boxes to a tank 4 ft. 8 in. by 3 ft. 6 in. having a canvas filter-bottom. A vacuum is maintained under the filter, obtaining a product containing from 30 to 60% moisture.

Supplies for November were as follows:

	Pounds.
Cyanide	4.34
Zinc	0.84
Lime	6.43
Pebbles and lining	2.31

Costs for November were as follows:

Labor: Crushing	$0.18	
Fine grinding	0.06	
Classifying	0.02	
Agitation	0.03	
Filtering	0.06	
Precipitating	0.04	
Miscellaneous	0.04	
		$0.43
Supplies		1.22
Power		0.07
Assaying		0.04
General, office, etc		0.09
Total		$1.85

(February 18, 1911)

The Editor:

Sir—In your issue of January 28, 1911, appeared an article by S. F. Shaw entitled 'Recent Cyanide Practice at the Montezuma, Costa Rica.' In this article Mr. Shaw speaks of the clogging of the spiral feed and also says that this feed throws out sand from time to time. In publishing this statement we are inadvertently done an injury. The explanation of Mr. Shaw's trouble is quite simple. He is feeding the material entirely too thick. We have never

heard of a tube-mill being fed with ore containing less than 37% moisture and do not understand how he is able to do any satisfactory work with the mills under the circumstances. In most plants the solution is 1 : 1, but, of course, it varies on different materials. If Mr. Shaw will have his material fed to the mill with at least 40% moisture he will overcome the difficulty and may also possibly increase the grinding capacity. Furthermore, if the mixture is of the proper consistence he can pump the mill two-thirds or three-fourths full and in that way partly balance the machine and reduce the horse-power required.

ABBÉ ENGINEERING COMPANY.

New York, February 7.

EMPIRE MINES CYANIDE PLANT

By Frank C. Landguth

(February 11, 1911)

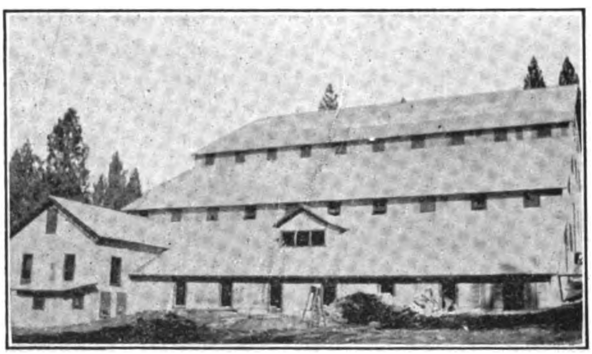
The Empire mines at Grass Valley, California, G. W. Starr, general manager, put in operation on December 1 a new cyanide plant, one of the most modern in the West. It was designed by Henry Hanson of San Francisco, assisted by myself. Mr. Hanson also built the plant, being assisted by J. T. Hooper. The plant has a capacity of 150 tons per 24 hours. The building is of wood covered with corrugated iron, the framework on the inside is painted white, and all the tanks and pipings are painted dark green. Electricity is used throughout for light and power, the current being delivered to motors six in number, ranging from 3 to 25 hp., at an energy of 440 volts. The site is an ideal one, as it permits the handling of all material by gravity.

The tailing from the stamp-mill flows through 425 ft. of 15-in. terra-cotta tile at 3% grade, supported on T-rail trestle, to the cyanide plant, where it is received direct by the Merrill classification equipment consisting of two settling-cones 8 ft. 4 in. diam., with sides sloping 50°, and five sizing-cones 4 ft. 8 in. diam., sides sloping 70°. To the apex of settling-cones is bolted a multiple casting tapped for 1½-in. pipe. Five of these pipes are in use, one feeding each sizing-cone. Each sizing-cone is fitted with Merrill hydraulic sizer. The pulp is here divided into two products, sand and slime. The slime overflow from settlers and sizers flows through an 8 by 12-in. galvanized-iron V-launder to a 3 by 4-ft. galvanized-iron sump filled with adjusting gates to give an equal distribution of slime to clarifiers, four in number, 24 ft. in diameter by 22 ft. deep, filled with false conical bottoms sloping 45° toward centre, and annular inside launders for clear-water overflow. This overflow is conveyed in galvanized-iron V-launder to a 20 by 16-ft. water-wash tank, from which it is drawn for various purposes.

The thickened slime, sp. gr. 1.4 to 1.8 is drawn from centre of the bottom of the clarifiers continuously through a 1½-in. pipe

which discharges 9 ft. above the bottom of the tank through a special cock, fitted with chilled-iron bushing, to a 6 by 10-in. galvanized-iron V-launder which in turn conveys sludge to four agitating-tanks 10 ft. diam. by 18 ft. deep, fitted with central cylinder special nozzles for compressed air at 10 lb. pressure, conical false-bottoms sloping 50° toward centre of tank and piped throughout for continuous system of slime treatment. After agitation the slime flows to the filter department, where it is treated by two Oliver continuous filters of 70-ton capacity.

Among the new features of the plant is the vacuum pump used in connection with Oliver filters which at 90 r.p.m. makes a 24-in. vacuum and elevates the filter effluent 41 ft. to a 16 by 20-ft, storage tank. This pump is doing excellent work and was designed by Edwin Letts Oliver. An Ingersoll Rand 10 by 10-in. compressor, making 150 r.p.m. furnishes compressed air for blowing slime-cake of the filters, agitating sludge, and aerating the sand charges.



CYANIDATION PLANT, EMPIRE MINES

The underflow of classified sand from sizing-cones is conveyed through a 10 by 10-in. wooden launder to the leaching department, where an automatic distributer charges the leaching-tanks, four in number, 24 by 10 ft., fitted with an annular inside overflow launder to carry overflow during filling, and a wooden filter-bottom of new design. These bottoms slope ¼ in. per foot toward the centre of the tank and are covered with the usual filter medium. The object in sloping these bottoms is to facilitate discharging through a 12-in. centre gate, Merrill type, wtih automatic sluicing machines. The sand residue is sluiced to waste through an 18-in. terra-cotta tile. The effluents from sand-vats are conveyed to sump-tanks in two 4-in. pipe-lines. These pipes are fitted with adjustable nipples so that effluent may be turned to any one of the four 20 by 10-ft. sumps. Adjacent to these sump-tanks are two automatic zinc-belts which feed dry zinc dust to a mixing-cone where a zinc emulsion is formed by adding a small amount of barren solution. This emulsion is conveyed to the pump suctions of two 5 by 5-in.

Aldrich triplex pumps, specially designed for handling cyanide solutions, which elevate the solution to two Merrill precipitating-presses of 250 tons capacity. Precipitated solution from presses flows to a 20 by 16-ft. storage tank. So far results show that the plant will be very efficient. I hope at an early date to be able to give a detailed description of the metallurgy.

CYANIDE PLANT AT THE EMPIRE MINES, GRASS VALLEY

By FRANK A. VESTAL

(November 9, 1912)

This plant was started on December 1, 1910, and has since been in continuous operation, treating the tailing and concentrate from a 40-stamp mill, together with the concentrate purchased from the Pennsylvania Mines Syndicate. The stamps used weigh 1050 lb. each, crushing approximately 3½ tons per stamp per day through a 35-mesh slotted screen. Both inside and outside amalgamation are used, 90% extraction being obtained.

From the plates the pulp passes over sixteen 6-ft. Frue vanners, which remove from two to three tons of concentrate per day. The tailing goes directly to the cyanide plant, while the concentrate is re-ground in a 4 by 8-ft. Allis-Chalmers tube-mill. The concentrate is shoveled into an 8-ft. receiving box of one ton capacity, which rests just above the mouth of the tube-mill at an angle of 6°. After passing through the tube-mill the product is raised by a water ejector to a Merrill concentrating cone, the overflow passing over two amalgamated plates 2 by 5 ft., where from 15 to 20% is saved. The oversize drops into a long receiving launder. This launder is movable, being attached to an 8-ft. screw shaft by means of a two-piece nut. The shaft is geared so as to move the launder 8 ft., the length of the receiving-box, in 6 hours. The discharge end of the launder is started over the lower end of the receiving-box, and the underflow from the cone washes the concentrate into the tube-mill at the rate of one ton in six hours.

The tube-mill is run continuously, handling about 4 tons in 24 hours, 85% of which passes a 200-mesh screen. The value of the concentrate is from $60 to $115 per ton. Much better work is being done in the leaching plant since the Pennsylvania concentrate has been added, as the tube-mill was not before run continuously and the concentrate formed in layers in the sand-vats, making the sampling of the residue very uncertain. The tailing from the vanners is conveyed to the cyanide plant through a 16-in. terra-cotta pipe. After 18 months' continuous use this pipe shows little or no wear. The concentrate, after leaving the tube-mill plates, is conveyed to the regular tailing launder in a 2-in. galvanized iron pipe.

The mill tailing is 46% fine and 54% coarse. The entire discharge runs to a distributing sump, which in turn feeds to two Merrill

settling cones, the overflow from these cones passing to four clarifiers or de-waterers, the underflow passing to four Merrill hydraulic sizing-cones. The overflow from these cones passes to the clarifying tanks, while the underflow goes to the sand-vats. The settling-cones are fitted with interchangeable nozzles, by means of which the overflow can be regulated as to amount and fineness. The sizing cones are fitted in the same way, but in addition water is introduced through a special casting at the bottom. This gives a decidedly flexible classifier, almost any product being obtainable either as sand or slime. The slime runs to a central distributing sump fitted with slide gates, which permits a varying amount being sent to any clarifier. These clarifiers are 22 ft. deep and 24 ft. in diameter, built with a false conical bottom. The centre cylinders in the clarifiers were originally 10½ ft. long, but on experimenting it was found that a cylinder 6½ or 7 ft. long would cause no agitation on the surface and give more settling room when forced to shut

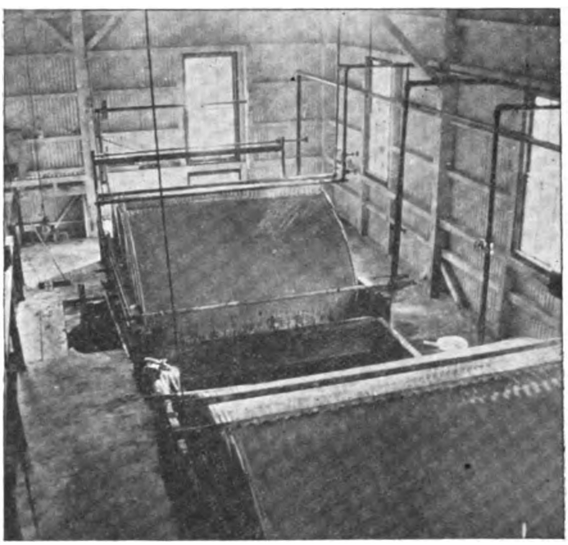
EMPIRE CYANIDE PLANT, GRASS VALLEY

down any part of the slime department for repairs, and still maintain a clear overflow.

The discharge is in the centre of the bottom, the pipe running to the side of the tank and rising 9 ft., allowing the slime to run to the Pachuca tanks by gravity. The discharge pipes are fitted with interchangeable Merrill nozzles, and any density of pulp is obtainable. At present pulp from 1.4 to 1.6 gravity is in use. There are four Pachuca tanks, 10 by 18 ft., run in series. The pulp is run into Pachuch No. 1, dropping into a 12-in. cylinder 8 ft. long; this is used to force the pulp to make at least one circuit before any part of it runs over into Pachuca No. 2. This Pachuca is fitted up in the regular way with the centre cylinder 10 in. from the bottom of the tank and a few inches above the level of the pulp. The discharge is about 4 ft. from the top of the tank and enters Pachuca No. 2, 10 ft. from the top, No. 2, 3, and 4 Pachucas being piped in the same way.

In No. 2 Pachuca the cylinder is only 8 ft. long, being submerged at all times. A baffle-board is fixed about 1 ft. from the top of the cylinder, the idea being to break up any possible heavy slime and to help in mixing. No. 3 Pachuca is arranged in the same way, with the exception of the cylinder, which is 12 ft. long. No. 4 Pachuca contains no central cylinder at all, being merely a tank with a false conical bottom. The pulp is fed directly to the filter-boxes from the No. 4 Pachuca.

Both cyanide and lime, dissolved with barren solution, are added to Pachuca No. 1 to get the necessary strength. Barren solution is also added to No. 2 and No. 3 Pachuca to bring the pulp to the consistence best suited to the filters. This solution, being added to No. 2 and No. 3 Pachuca, freshens up the charge and gives



OLIVER FILTERS

better results than when all added at No. 1. Sufficient lime and cyanide are added to keep effluent solution from filters at a strength of 0.04% KCN and a sufficient degree of protective alkalinity.

The cyanide consumption ranges from 0.75 to 0.90 lb. per ton of ore. Several experiments were made with solutions ranging from 0.04 to 0.12%, but the extraction was no better, while the cyanide consumption increased almost at the same rate as the amount of cyanide added. In other words, using a 0.12% solution, the consumption reached 0.09%, or 1.8 pounds.

The pulp passes to two 35-ton Oliver continuous filters. The gravity is kept between 1.3 and 1.35.

An Oliver wet-vacuum pump is used, maintaining a vacuum of 22 in. The filters are fitted with a front and back wash, being

piped so as to permit either barren solution or water being added either in front or back or both. When the filters were installed it was thought that a front wash would be all that was necessary for this particular product. Some experiments made may be of interest.

Before the concentrate was added, the slime-heads averaged in precious-metal content as follows:

January, heads \$1.81, residue 24c.; April, heads \$1.99, residue 25c.; February, heads \$2.03, residue 30c. This shows the residues without a back wash.

May, heads \$1.95, residue 14c.; June, heads \$2.11, residue 15c.; July, heads \$1.65, residue 11c. With both front and back wash.

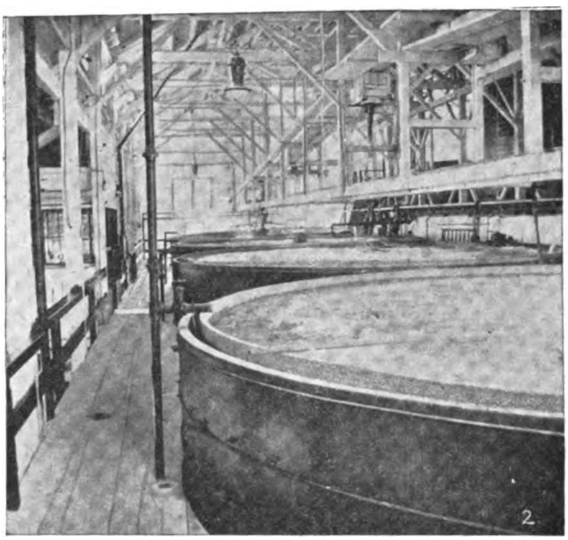
After the sulphides were added the gold content increased as follows: August, heads \$2.38, residue 17c.; September, heads \$2.89, residue 22c.; November, heads \$2.60, residue 20c. With both front and back wash.

On a 2-hr. test made without a back wash the residue content reached as high as 40c. per ton. The filters, making a revolution in 8 minutes, allow 1¼ min. for washing with a front wash. With the added back wash a period of 2¾ min. is obtainable. The former time is more than doubled because on the front side of the filter one compartment (or 1/24 of the filtering surface) is discharging the cake, while at the back side that one compartment is gained for washing.

To prevent the excess water or barren solution added at the back of the filter from running down into the filter-box and diluting the solution and thinning the pulp, a launder was constructed to catch this excess. In the first experiment a heavy piece of rubber belting formed in the shape of a U-launder and supported by a strip of wood across the filter-box, was set at an angle, the excess wash running to waste at the lower end. This was not satisfactory, mainly because a thick belt was necessary for strength, and as a result the water would strike the edge of the belt and be carried to one edge of the filter and deposited in the filter-box. Next a wooden V-shaped launder was built and a strip of rubber 1 ft. wide by 1/32 in. thick was tacked to the side of the launder. This rubber was allowed to rest against the cake for practically two-thirds of its width. This caught nearly all the excess for the first few days, but later it began to curl up and during the cold weather would freeze. Another objection was that when the power would go off, or a shut-down occur, on starting up the filters the vacuum would tear the rubber from the launder. Next a sheet-iron launder (No. 20 iron being used) was made with one side 15 in. high and the other 3 in. The high side is bent to meet the contour of the filter, with the upper edge flattened out so as to only leave a bearing surface of ½ in. The overbalancing of the launder against the filter holds it in place. Up to the present time this launder has given entire satisfaction. In 24 hours hardly enough fine slime is washed away for an assay, and the excess water only carries a trace of gold.

The filters are doing good work and require little attention, only one man being required to run the entire plant on the two night shifts. A cake from 3/16 to 5/16 in. is formed. The slime is practically free from granular particles, and is exceedingly sticky and impervious. The scraper for removing the discharge cake must be filled at least twice a week, taking one man about 15 minutes for each. When properly taken care of, the life of the canvas is from five to six months. The Pachuca tanks are run in series and several samples taken at different times between the tanks showed a uniform extraction, proving that very little, if any, slime made the circuit before being attacked by the solutions.

There are four sand-vats of 155 tons capacity each, fitted with inside overflow-launders. The sand is carried by means of an



LEACHING VATS

overhead launder to each vat and distributed by a Merrill distributor. During the filling, lime is added to the charge, about 2 lb. per ton of ore being used. After filling, the charge is allowed to drain for 5 hr., the top being raked and leveled and 100 lb. of slaked lime added. After a vat has been filling for 12 hr. the drain valve is opened, which, in decreasing the overflow, lessens the chances of losing gold in this way. The gold being mostly in the fine, makes this necessary. Several samples were taken of the overflow water, the amount of solids being weighed and assayed, and it was found that from 1/6 to ¼ ton was being lost every 24 hr. This assayed $2 per ton, a maximum loss of 50c. per day. As an experiment an inner rim 12 in. high was placed inside of the regular overflow edge of a vat, this rim resting 1 in. higher than the top of the vat, and being held away from the inside of the regular overflow

by small blocks ½ in. thick; the idea was to get away from the waves caused by the distributor and to force the overflow to go down under the inside rim. A much more even overflow was thus obtained, but several samples taken showed little, if any, saving over the old method.

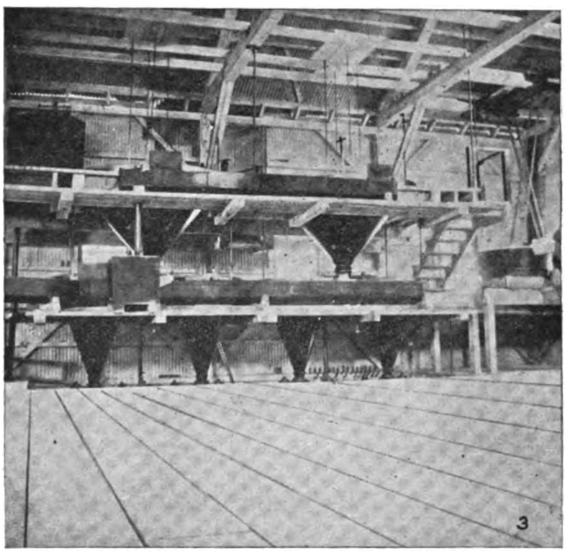
The product in the sand charges is 87½ coarse, against 12½% fine. After the 5-hr. drain period, a 0.10% solution is added to the top of the charge, 40 tons in all being used. The solution coming from the vats is allowed to go to waste until it shows a trace of KCN, when it is turned into the gold sump. After a KCN strength of 0.04% has been reached, the content is the highest. At this time the effluent valve is closed and the charge is allowed to stand for 10 hr.; the idea is to give less solution tonnage and at the same time enrich the solutions. The effluent solution from the filter is added as a second solution. A barren solution would be preferable for a second solution, but it is absolutely necessary to clarify the filter-solution before precipitation takes place, and the sand-vats are used for this purpose even after the extraction has been completed. A small clarifying tank as well as an excelsior filter is used in the event that the sand-vats cannot handle the filter output, which amounts to 144 tons per 24 hr. Either barren solution or water is used for the final wash, about 70 tons being necessary.

Each vat is fitted with a Merrill centre gate and has a sloping bottom to facilitate sluicing. The vat itself is built in the regular way, but the false-bottom is 8 in. high at the outside against 2 in. at the centre gate. An automatic sluicing machine is used for discharging the vats. This is similar in construction to a lawn sprinkler, hanging from an overhead track. This type of machine was first used, I believe, by E. L. Oliver at the North Star mines. It works well, but plenty of water is needed. A hose is necessary for the last half-hour's sluicing for cleaning up the bottom. The weaker solutions (both in gold and KCN) are precipitated separately, and later run to waste if the assay returns are satisfactory. A great portion of this, as stated before, is used as a wash on the filter or sand-vats, or for thinning the pulp in the Pachuca tanks.

The well known Merrill zinc dust precipitation is used. Two 125-ton presses, with ten 36-in. frames, handle the total solution tonnage for the month. The zinc dust is uniformly distributed along a rubber-belt conveyor of the same length as the depth of the tank. The belt is operated by means of a large float which, sinking with fall of the solution being pumped, moves the belt at the proper speed. The zinc falls into a receiving-cone, where barren solution is added, an overflow pipe carrying the zinc emulsion to the suction pipe of the pump. A mechanical agitator is used in the cone in preference to air. The air tended to oxidize the zinc and at times gave more or less trouble with precipitation. In my opinion the belt feeder arrangement gives more uniform speed than any other feed if properly taken care of. The zinc consumption amounts to about 0.19 lb. per ton of ore. Consumption per ton of solution has

averaged as low as 0.12 lb.; at times 0.10 lb. of zinc per ton of solution has been used, giving 1c. barren solution. The weak solution is kept at a strength of not less than 0.03% KCN. A less amount of zinc is added to the higher-grade solution, as it is standardized and used as strong solution. Using less zinc gives a higher-grade precipitate and makes a saving in several ways, while 4c. or 5c. barren solutions are of no importance when the solution is saved.

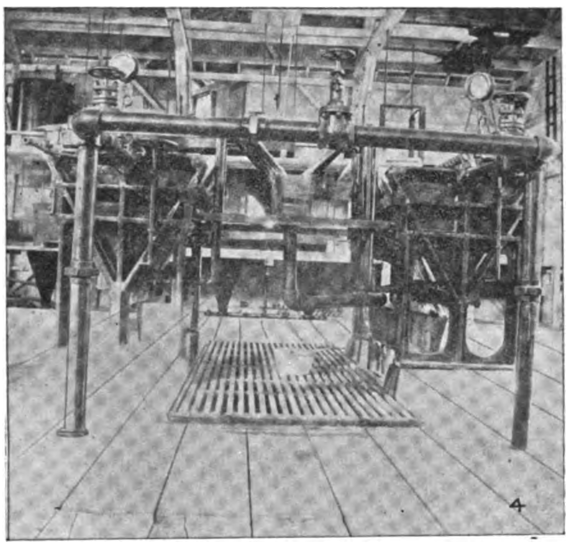
More or less trouble was encountered during December and January. At first it was thought that the cold solutions were causing this, the temperature being around 36°F., but several tanks were heated, with no better results. Forty tons were pumped with temperature at 37°, and the remaining 40 tons of the same tank heated to 56°, and the results were the same, although a little more zinc



CONE CLASSIFIERS AND LIME-FEEDER

was necessary with the cold solution. At this time the sand treatment was changed in such a way that the vats could not handle the filter solution, so the other small filters, for clarifying purposes, were taxed to their capacity. Apparently the pregnant solutions were perfectly clear, but on close examinations were found to contain a little fine slime. On opening the presses a thin layer of slime was found at the top of each frame. On changing the treatment of the sand and cutting out the filters, good results were almost immediate. Excess zinc tended to relieve matters a little, and this led to the belief that the fine slime merely coated over a certain part of the zinc being added and also that already in the press. While our precipitation troubles have been few, they have invariably been traced to solutions not having been thoroughly clarified.

Continuous precipitation has proved more satisfactory than intermittent. The precipitate as it comes from the press is worth from $18 to $30 per pound. The precipitate is melted with a flux containing ½ of borax to 1 of dry precipitate, 20% soda, 2% silica, which gives the best results. One No. 40 and one No. 120 Steele-Harvey tilting furnace is used, with crude oil as fuel. Originally the precipitate was dried to just before the dusting point, and then screened and fluxed; at present it is not dried nor screened, but is fluxed and melted direct from the presses. In January one-half the total product, which contained 49% moisture, was fluxed and melted in less time than formerly required for melting the nearly dry product. It is necessary, of course, to feed in the precipitate fast enough to keep a crust on the top of the metal or 'spitting' will



MERRILL PRECIPITATION PRESSES

result. When pouring, the pot is emptied each time, allowing the next charge to be added at once with no chance of spitting.

The zinc fume from the furnaces was objectionable, and it became necessary to use some means of controlling it in order to protect the men. The first thing tried was a large inverted cone with a stack leading up through the roof (the building itself is fitted with a cupola for taking away the smoke), which resembled in every respect the stack on a blacksmith forge. This was not entirely satisfactory and was the cause of some dusting, the draft being very strong. A device for catching the dust was arranged in the stack of the large furnace, with the result that the greatest amount caught was 4 lb. per run, the value being $2 per pound. At present a sheet-iron house is placed around each furnace, being built from the floor up and connected to a flue 20 in. square, which

runs on a downward angle for a distance of 25 ft. At this point a T was put in, the bottom being fitted with a sliding door and acting as a dust-chamber. The upper part runs into a stack of suitable height. Doors were put in at the proper places around the furnace and glass panels arranged to the best advantage for watching the fire. This arrangement has proved very satisfactory, no trouble arising from the fume. Practically no dusting takes place, none showing at the top of the stack and little being deposited in the dust-chamber after three melts.

The slag carries more gold than if other methods of melting were used. At certain periods the slag is sacked and shipped to the smelter. Great difficulty is met in getting slag assays to check, owing to the shot. In sampling, every ninth shovelful is reserved and run through a small crusher. This product is again cut down and crushed to a fineness of about 40 mesh. This is panned and the shot separated; the percentage of shot is recorded and assayed, the rest of the sample being reground and assayed. In this way a fair check is made on the amount that should be in the slag, using the solution and precipitate value as a check against the bullion and slag. At present experiments are being run in treating the slag in the mill grinding-pan. One hundred and fifty pounds of slag was run through a small crusher and an assay taken showing a value of $1 per pound. This was fed into the grinding pan and silver added in the necessary quantity. The result was 90 oz. of a mushy amalgam which retorted 30 oz. of base metal. This was in turn melted down (after adding litharge to give uniformity for sampling), and this assayed 347 fine. The results were $215.10 from a supposed content of $250, still leaving 14c. per pound in the slag. Assays taken from the slag residue run $0.075 per pound. If future tests work out as well, the entire product will be treated in the same way, as the cost is small and the returns are quick.

The treatment cost per ton of ore, averaged for six months, after adding concentrate showed an increase over the preceding six months of $0.005 to $0.02.

Extra Cost Due to Treating Concentrate

Lime to tube-mill, ½ ton	$ 8.00
Extra lime to cyanide plant, 1 ton	16.00
Extra KCN to cyanide plant, 100 lb	18.90
Extra power	15.00
Extra pebbles	6.50
Total	$64.40

Taking the average tonnage per month makes the cost about 65c. per ton. While this seems small, a careful examination of the past 10 months' run shows this to be a just charge. An extraction of from 87 to 89% is obtained from the sand, while the extraction on the slime is 90 to 93%. Pipes are being laid direct from the tube-mill plates to the Pachuca tanks with the idea of treating all the concentrate with the slime.

The plant contains two Aldrich triplex solution pumps run by two direct-driven 3-hp. motors; one Oliver wet-vacuum pump, run by a 15-hp. belt-driven motor; two Oliver filters, driven by a 7½-hp. motor, belt driven; a 4-ton Allis-Chalmers tube-mill, 10-hp. motor, belt driven; and one lime-pan with a 2½-hp. motor. The actual horse-power used per motor amounts to:

One solution pump	1.6
Compressor	15
Vacuum-pump	8
Filter	2
Tube-mill	8
Lime-pan	0.7

One shiftman and one helper are employed on day-shift, and one shiftman on each of the other two shifts. A foreman has charge of the cyanide, refining, and assay office, remaining on day-shift all the time. The assayer does the work for the mine, mill, and cyanide plant. The treatment cost for 1911 amounted to:

Labor	$0.14
Power	0.035
KCN	0.17
Lime	0.03
Zinc	0.02
Assay	0.03
Refining	0.02
Total	$0.445

Henry Hansen was the metallurgist and designer of the plant and J. T. Hooper was superintendent of construction, with F. C. Languth as metallurgical engineer on the ground to put the plant in operation.

COLORADO-TOLEDO MILL

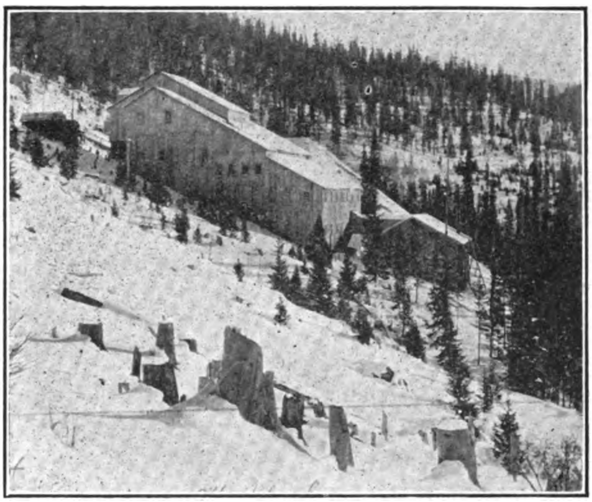
By W. B. Le Wald

(February 18, 1911)

The Colorado-Toledo Mining Co., which is developing mines on Collier mountain in Summit county, Colorado, is completing a mill that is to be put in operation early in March. The ores to be milled consist of mixed sulphides containing lead, zinc, gold, and silver. Mining is done by means of an adit, now driven 3000 ft. into the mountain and tapping veins at 1000 to 2200 ft. in depth. About 1200 ft. of driving has been done on the various veins, and the adit is still to be extended 1400 ft. to intersect additional veins. The scheme of treatment is given below.

After weighing on scales the ore from the mine is delivered in cars to two bins of a total capacity of 135 tons, situated outside the mill, and so placed that they may receive the mine ore and store it when the crusher is idle. From these bins the ore is fed to a No. 5 Austin gyratory crusher. This is of large capacity, and is designed to crush enough ore in 8 or 10 hours to supply the mill for

24. One man on an 8 or 10-hr. shift will therefore crush all the ore, thus doing away with labor in the crushing department for the remainder of the 24 hr. From the crusher the ore is delivered automatically to a belt-elevator with 14 by 6-in. buckets running at about 400 ft. per minute, and is elevated to a revolving screen or trommel fitted with a No. 7 wire screen with 1¼-in. openings. The undersize from this screen (1¼ in. or about 26 mm. and finer) is automatically sampled and sent to the mill-bins. These have a total capacity of 270 tons. The oversize from the screen will be returned to the crusher. It is a question whether by returning the oversize to the crusher the ore will be broken fine enough, but it is thought that with a 'choke' feed it will. If so, there will be the saving of a small crusher for the re-crushing of this oversize. The



COLORADO-TOLEDO MILL, MONTEZUMA, COLORADO

crushing department is operated by a 35-hp. motor, and is entirely separate and distinct from the main mill.

From the mill-bins the ore is fed by automatic-plunger feeders to a 20-in. belt-conveyor which delivers it to a bucket elevator, with 18 by 6-in. buckets running at 400 ft. per minute. This elevator delivers to the revolving-screen line which consists of five trommels, three 6 ft. long by 3 ft. diam., and two 9 ft. long by 3 ft. diam. The short ones are for the coarser sizes, and the longer ones for the fine sizes. The wire used will be as follows:

Mesh.	No. Wire.	Opening, mm.
2½	13	8
3	10	5
4	11	3¼
6	13	2
12	18	1

The oversize from the 8-mm. trommel will be sent to coarse rolls, re-ground to 8-mm. size, and returned to the trommel. The 8 to 5-mm. size will be delivered to No. 2 jigs, the 5 to 3¼-mm. size to No. 3 jigs, and the 2 to 1-mm. size to No. 4 jigs. The ore 1-mm. and finer goes to a 4-compartment hydraulic classifier where the No. 1 spigot delivers to No. 5 jigs. Spigots No. 2, 3, 4, and the overflow, go to Card tables. The No. 1 jig is two-compart, treating 8 to 5-mm. size, and will collect clean galena on the first compartment and impure galena on the second. The tailing will be sent to the No. 2 or fine rolls for re-grinding. The re-ground product is to be returned by the elevator to the trommel line. The No. 2 jig is also two-compartment, treating 5 to 3¼-mm. size. Its products are the same as No. 1 jig and the tailing is treated in the same way. The No. 3 jigs are four-compartment, treating 4¼ to 2-mm. size, and will make clean galena on the first compartment, galena and iron on the second, iron-zinc middling on the third, and zinc on the fourth. The tailing goes to the fine or No. 2 rolls, as in No. 1 and No. 2 jigs. The No. 4 jigs are four-compartment, treating 2 to 1-mm. size, and making similar products to the No. 3 jigs, and tailing going to the tailing launder. The No. 5 jigs are four-compartment, treating the discharge from the first spigot of the hydraulic classifier, the product and tailing being similar to that from the No. 4 jigs.

Of the above-mentioned jig products, the galena and the galena-iron will be a concentrate for shipment to the smelter. The iron-zinc middling is re-ground for further treatment. The zinc may or may not be marketable, depending on ore conditions, and how perfectly it can be separated from the other constituents. The bed-screens on the jigs will be as follows: On No. 1 and 2, 4-mesh, No. 11 wire, giving a 3¼-mm. opening; on the No. 3 and 4, 6-mesh, No. 13 wire, giving a 12-mm. opening. The same wire will be used on the No. 5, but this jig will be bedded with concentrate from the No. 3 jig. Space has been reserved so that if it prove necessary, machinery can be set for re-grinding and treating separately the zinc middling. As now arranged, the middling goes to the hydraulic classifiers, and the first spigot product is sent to the No. 5 jig. The No. 2 spigot is delivered to coarse concentrating tables, the No. 3 spigot to other tables for medium-size, and No. 4 spigot to fine tables. All these tables will make a galena-iron and iron-zinc middling, a zinc product, and a tailing. The products will be similar to the ones made on the jigs and be mixed with them for shipment or other disposition.

Arrangements have been made to dewater all jig tailing and concentrate, excess water from the tables being settled in large tanks and the clear water decanted and sent to waste. The thickened settling is to be delivered to a Card slimer, which will make the same products as the other tables. All concentrate will be delivered to bins by gravity. Provision is made for automatically sampling the tailing or waste from the mill, and all concentrate. The mill is heated, and driers for concentrate provided. Water for

mill operation is distributed from an 8500-gal. tank, fed by a 6-in. pipe, and a fire-line, separate from the operating line, and with full main pressure, has been provided. It is expected to operate the mill with two men on a shift, with the addition of a crusherman for the 8 or 10-hr. shift.

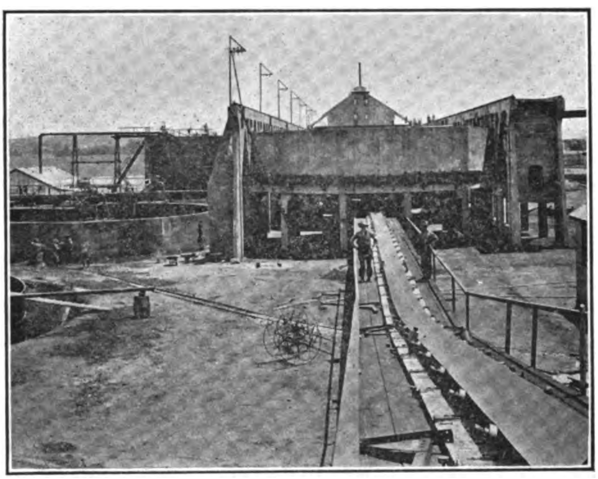
MODERN REDUCTION PLANT ON THE RAND

By Rowland Gascoyne

(February 25, 1911)

There has just been started at the City Deep mine on the Rand a reduction plant possessing several unusual points of interest. This plant has been previously described, but its many new features entitle it to more than ordinary attention.

Timber, in the past, has entered largely into the structure of

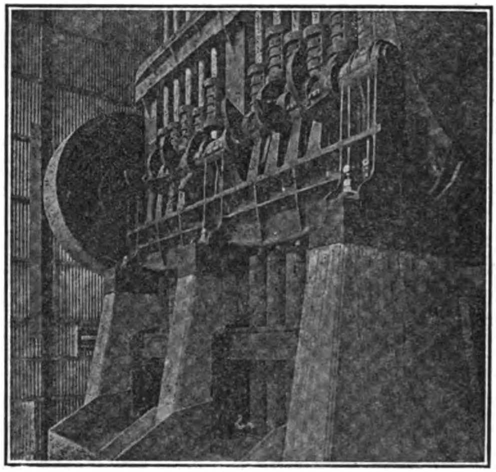


CONVEYOR BELT AND CYANIDE VATS

mills on the Rand, but in this instance timber is almost conspicuously absent, its place having been taken by reinforced concrete, and steel girders, and other forms of steel construction. There is, of course, the risk in displacing timber by girders in a mill, that a good deal of shearing of rivets may take place; and again, steel and reinforced concrete can scarcely be expected to have the same accommodating results as timber. In the 600-stamp mill at Randfontein, on the other hand, no effort to spare the use of timber seems to have been made, and reinforced concrete has been only sparingly used.

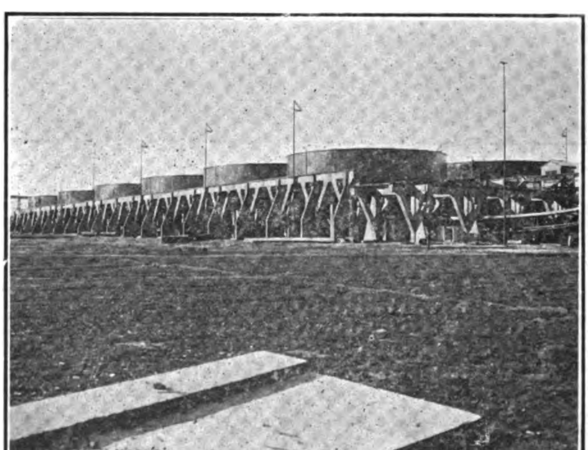
The plant is designed to treat 65,000 tons per month, and as the ore is of higher grade than the average of the neighboring mines, when once it gets into proper stride the mine will rank as one of the leading producers on the Rand. Saving of labor costs and economy of handling are other prominent features, while a high percentage

of extraction is regarded as certain of attainment. As all the gold will be recovered at one spot in the gold-recovery house, the possibility of thefts, loss from which is estimated to run in some cases up to 10% of the output, ought to be reduced to a minimum. The



DETAIL OF STAMPS

mill will be driven, as in fact will all the machines at the surface of the mine, by purchased power from the Victoria Falls Power Co., as soon as that company is in a position to supply all the motive

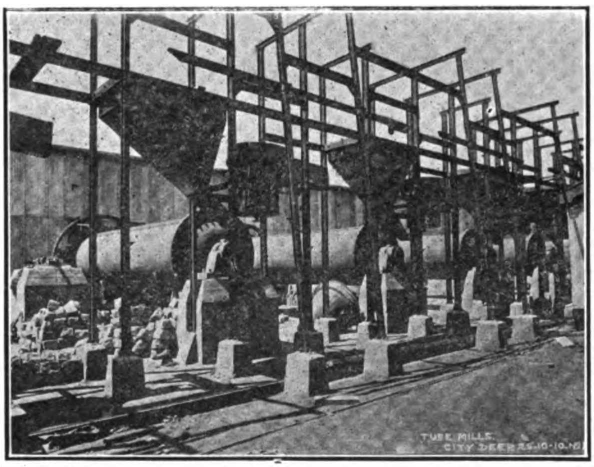


COLLECTING SAND VATS

power required. At the present time, 50 stamps out of the 200 are running. Power is carried to the required points by underground cables.

The ore is sorted and crushed at the mine in the ordinary manner before being conveyed to the mill. The transport of ore to the mill and all necessary surface work is effected by heavy electrical locomotives using 2000-volt, 50-cycle, 3-phase current.

The stamps are arranged in units of 10, the weight of the stamps being 2000 lb., and the stamps having long heads and short stems. One special feature is that there is only a layer of ½-in. felt between the mortar bases and concrete foundations. Kingposts are entirely dispensed with, the concrete foundations being carried, with indented steel-bar re-inforcing, to above the level of the top of the mortar-box. Each cam-shaft is carried by a steel frame, and rests on eleven bearings, so as to minimize the risk of breakage. Stems are 4 in. by 13 ft. long, and the stamps are arranged for a very heavy duty if necessary. Each battery is provided with four Challenge feeders.



TUBE-MILLS

Adjoining the mill are nine 22 by 5½-ft. tube-mills, each driven by a 100-hp., slow-speed motor through a reduction-gear; all the other motors are driven by belts.

Next to the tube-mill house, and between that and the sand-plant, the gold-recovery house is placed, and under the same roof are arranged the amalgamating tables, extractor-boxes, clean-up machinery, strong-room, and refinery, as well as the office of the man in charge of the ore reduction.

The solution sumps are ample in size to meet future extensions of the plant and are placed close to the gold-recovery house. They consist of three sumps 48 ft. by 47 ft. by 12 ft. 9 in. deep, the slime solution sump being 122 ft. by 47 ft. and 12 ft. 9 in. in depth.

The collecting sand-plant consists of a row of 6 vats, 50 ft. diam. and 10 ft. deep, on reinforced concrete supports. A 24-in.

belt conveys the sand to the top of the sand-leaching vats, consisting of two rows of 6 vats, each 56 ft. diam. by 12 ft. deep, built of steel and carried on supports of reinforced concrete.

The slime-plant consists of four conical-bottom collectors, 60 ft. by 10 ft. and 16 ft. in depth, likewise on reinforced concrete supports. There are two steel, conical-bottomed, air-agitation vats, 32 ft. in diameter and 30 ft. deep, and 8 steel, conical, first and second-wash vats, 70 ft. in diameter.

The sand from the leaching vats is discharged by a Blaisdell excavator delivering to a 24-in. belt under each row of vats. The sand is conveyed by another 24-in. belt up an inclined steel cantilever frame, and is discharged 100 ft. above the level of the ground.

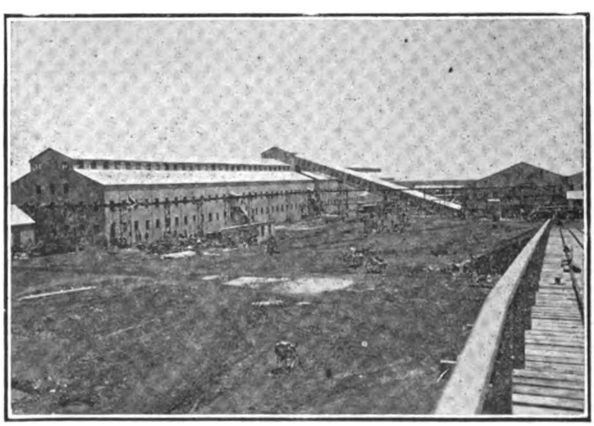
For some time past ore has been developed in the mine on ample scale. In fact, during the last twelve months, 856,879 tons of profitable ore has been developed, so that by the time the mill gets into full swing several years reserve will be in hand. The following ore has been developed during last year:

Quarter ended	Exposures: Distance, ft.	Width, in.	Value, dwt.	Pay-Ore: Quantity, tons.	Value, dwt.
December	2006	19.8	19.8	183,155	7.2
March	1471	19.9	26.0	177,659	10.7
June	3353	17.7	24.7	310,313	9.2
September	2506	17.8	22.2	185,752	7.3

NEW RANDFONTEIN MILL

(April 8, 1911)

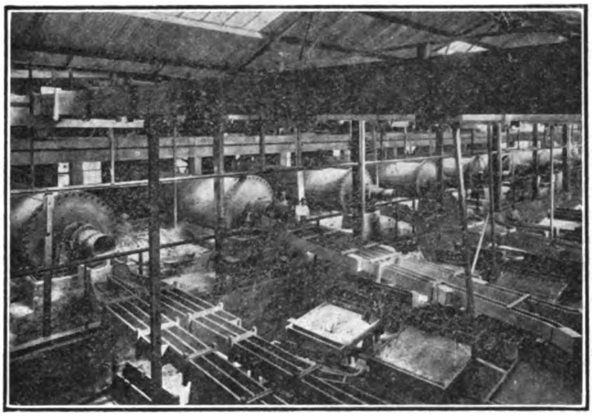
A recent occurence of note is the starting of 300 stamps in the huge mill recently erected on the Randfontein property, which



RANDFONTEIN CENTRAL MILL

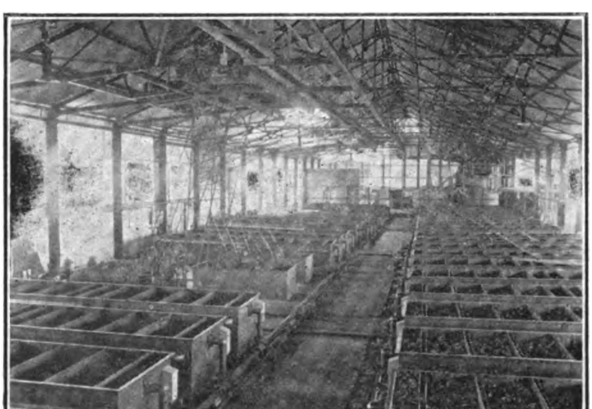
claims to be the biggest reduction plant under one roof in the world. The mill has 600 stamps, supplemented by 16 tube-mills,

and the necessary sand and slime plants. Taken altogether, it is an imposing structure. There are already 400 stamps on the Randfontein property, so that the new mill will bring the number



TUBE-MILLS. RANDFONTEIN CENTRAL

of stamps available up to a thousand. It will be readily understood that to keep all these stamps supplied the ten mines in operation on the property will be fully employed, and to bring the ore reserve up to the mark some effort will be required. At



GOLD-RECOVERY HOUSE. RANDFONTEIN CENTRAL

present the ore reserve represents two years milling, which according to modern Rand practice is below ordinary local requirements. When all the stamps get fully at work at Randfontein, it

CRUSHER HOUSE WITH CONVEYOR-BELTS FROM BIN TO CRUSHER

is confidently expected that the property will be not only the premier producer of Transvaal, but also of the world.

NEW ALVARADO MILL

(April 8, 1911)

The Alvarado Mining & Milling Co., owner of the Palmilla mine, situated three miles from Parral, Chihuahua, has completed and put in operation its 60-stamp mill and cyanide plant, and we give herein a brief description of the equipment and its arrangement. The mill and power-plant are separated from the mine by several thousand feet, requiring the use of a surface gravity tram-line, and an aerial tramway for conveying the ore from the mine to the mill. The ore is carried over the surface tramway in 5-ton cars which are discharged into loading-bins at the nearest terminal of the aerial line of buckets, the latter having the capacity to deliver ore at the mill-bins at the rate of 30 to 40 tons per hour. The ore is discharged by gravity from the receiving-bins, passing over six grizzlies to three 9 by 15-ft. Blake crushers, thence by plunger feeders to three 19-in. belt-conveyors, the latter delivering the crushed ore to three battery-bins, each of 800 tons capacity. The ore passes from these bins by gravity through Challenge feeders to the twelve 5-stamp batteries, the mortars being of the El Oro type, and the stamps of 1160 lb. each. Each battery is driven by a 30-hp. motor, and the stamps make 102 drops per minute. The mortars are set on a concrete foundation extending the length of the battery floor. The pulp discharged from the mortars is passed through cone-classifiers to Deister sand-tables, the tailing from the latter passing to four Dorr classifiers which separate sand from slime. The sand product of the Dorr machines is then pulverized in four 5 by 22-ft. tube-mills; the tube-mill pulp

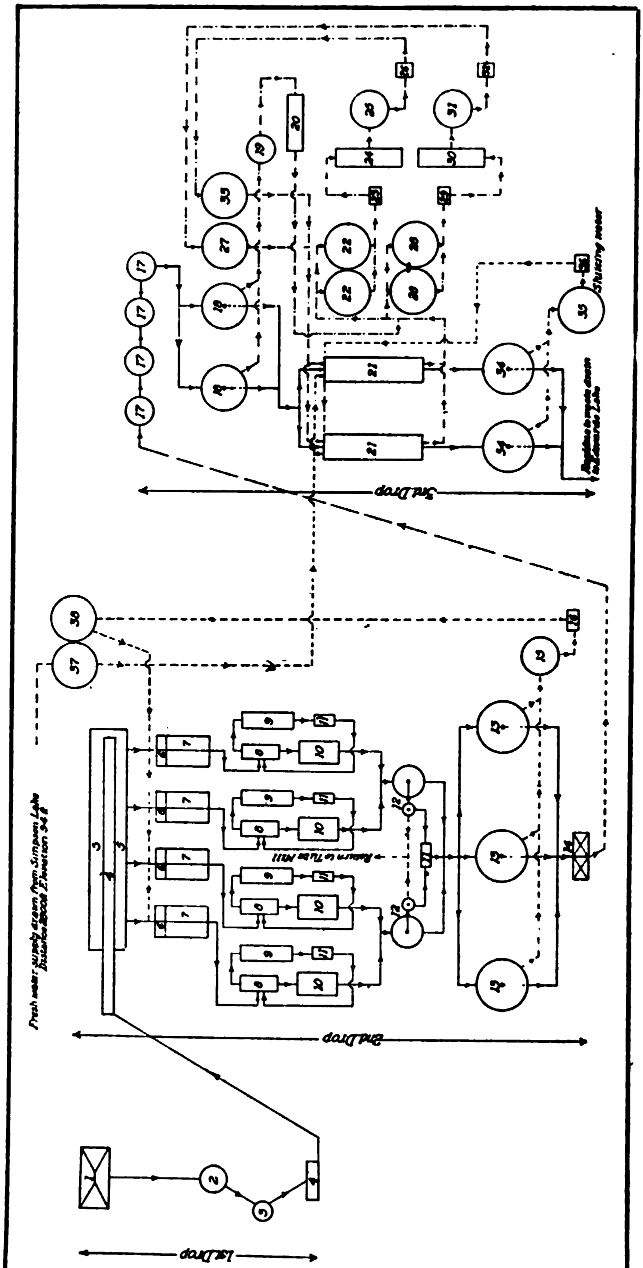
is carried by three 10-in. bucket elevators to three cone-classifiers, the oversize being returned to the Dorr machines for re-classification, the slime passing to Deister slime-tables, by which it is re-concentrated. The tailing from the slimer is then passed to six dewatering cones, each 16 by 18 ft., the clear water flowing to two 15 by 30-ft. tanks, and is pumped thence to storage-tanks for use in the batteries again. The slime from the dewatering cones is pumped to Pachuca tanks, of which there are 12, each being 15 ft. diam., and 47 ft. deep. After the aeration and cyanidation takes place in the Pachuca tanks, the slime is passed to Butters filters of 180 leaves, divided into two sections, each section having three hoppers, with a 13 by 13-in. Wheeler discharge valve for each hopper, through which the clear solution passes to the precipitating department.

The power-plant contains five Babcock & Wilcox water-tube boilers, each of 250 hp.; a Nordberg cross-compound, two-stage air-compressor, with the capacity to deliver 1800 cu. ft. of free air per minute; four cross-compound Corliss engines, three of which are direct-connected to that number of Allis-Chalmers generators. Each generator is a 3-phase, 480-volt, 600-ampere, 60-cycle machine. The electrical equipment, including transformers, providing for the delivery of power to all parts of the mill and to the Palmilla mine, is the most complete that can be devised. The Allis-Chalmers Co. supplied the generators and electrical equipment, while the Minneapolis Steel & Machinery Co. constructed the power-plant. The Alvarado M. & M. Co. is under the management of John I. Long of Parral. The mill was designed by Bernard MacDonald, consulting engineer for the company. The construction work of the mill was under direction of T. H. Gracey.

DOME MILL

(May 20, 1911)

On the next page is a flow-sheet of the 40-stamp mill which has been erected for the Dome Mines Co., by the Merrill Metallurgical Co. It is of especial interest as being the first, other than experimental mills, to be erected in the Porcupine district of Ontario. The ore so far found has proved easy to treat. The figures in the illustration correspond to machines and data as below: (1) hoist; (2) Kennedy No. 7½ gyratory crusher; (3) Kennedy No. 5 gyratory crusher; (4) belt-conveyor to mill-bins; (5) battery storage-bin; (6) four 10-stamp batteries, 1250-lb. stamps, 102 drops per minute; (7) primary amalgamating-plates, 54 by 144 in., slope 1½ in. per foot, plates in two sections; (8) four duplex Dorr classifiers; (9) four 5 by 22 A. C. tube-mills, El Oro lining, scoop feed, spiral discharge, 31 r.p.m.; (10) secondary amalgamating-plates, 108 by 144 in., slope ½ in. per foot, in two sections; (11) five Frenier pumps, 8 by 54 in.; (12) two concentrating-cone units; (13) three Dorr thickeners, 30 by 10 ft.; (14) duplex bucket-ele-



FLOW-SHEET, DOME MILL

vator, centre to centre 70 ft., buckets 7 by 16 in.; (15) return battery-water sump, 20 by 10 in.; (16) return battery-water pump, Aldrich vertical triplex, 350 gal. per minute, 15-hp. motor; (17) four continuous agitators 8 by 40 ft.; (18) two Dorr thickeners 25 by 10 ft.; (19) thickener overflow tank 10 by 10 ft.; Merrill clarifying-press; (21) two Merrill slime-presses, 76 four-inch frames, Dome type; (22) two strong-solution precipitating-vats, 25 by 10 ft.; (23) strong-solution precipitation pump, Aldrich vertical triplex, 175 gal. per min., 7½-hp. motor; (24) strong-solution (Merrill) precipitation-press, 52-in. press, 20 two-inch frames; (25) strong barren-solution sump 20 by 10 ft.; (26) strong barren-solution pump, Aldrich vertical triplex, 175 gal. per minute, 7½-hp. motor; (27) strong barren-solution storage-tank 25 by 10 ft.; (28) two weak-solution precipitating-vats, 25 by 10 ft.; (29) weak-solution precipitation-pump, Aldrich vertical triplex, 100 gal. per minute, 5-hp. motor; (30) weak-solution (Merrill) precipitation-press, 10 two-inch frames; (31) weak barren-solution sump, 20 by 10 ft.; (32) weak barren-solution pump, Aldrich vertical triplex, 100 gal. per minute, 5-hp. motor; (33) weak barren-solution storage-tank, 25 by 10 ft.; (34) two Dorr thickeners, 30 by 10 ft.; (35) sluicing-water sump, 25 by 10 ft.; (36) sluicing-water pump, Aldrich vertical triplex, 700 gal. per minute, 50-hp. motor; (37) fresh-water storage-tank, 25 by 20 ft.; (38) battery-water storage-tank, 25 by 10 ft.

OPERATION OF THE GOLDFIELD CONSOLIDATED MILL

By J. W. Hutchinson

(May 6, 13, 20, 27, June 10, 1911)

Construction

As this article will be confined strictly to the operations at the plant, those interested in general conditions at Goldfield, Nevada, are referred to J. R. Finlay's 'Cost of Mining,' T. A. Rickard's articles on Goldfield which appeared in the *Mining and Scientific Press* during 1908, and the report on the geology of the district by F. L. Ransome published by the U. S. Geological Survey. The details of construction of the 100-stamp mill and cyanide plant of this company have received publicity through Bulletin 1438 of the Allis-Chalmers Co., which is substantially correct. However, in this bulletin due credit is not given J. B. Fleming, of San Francisco, who was employed by the Goldfield Consolidated Mines Co. as mechanical engineer, and who drew up the specifications and general plans. On these specifications the Allis-Chalmers Co. bid and secured the contract for the crushing and transmission machinery and electrical apparatus. The engineers of the Allis-Chalmers Co. were not employed by the Mines company until the general plans were out and the contract had

been let. This is mentioned for the sole purpose of rendering to Mr. Fleming publicly the credit which this company gives him.

During the first year's operation, from December 26, 1908, to December 26, 1909, the stamps crushed through 12-mesh; the product, after classification, going to tube-mills. After the 20-stamp mill of this company, known as the Combination mill, was abandoned in September, 1909, it was decided to increase the capacity of the 100-stamp mill from 600 to 850 tons per day.

The installation of 40 additional stamps, three tube-mills, and 25 concentrators was at first considered. This would have largely increased the floor space required, the structural steel for building, and, because of the contour of the hill, enormous concrete foundations for ore-bins and battery-block would have been needed. In addition, the increased tonnage could not have been handled for six months. The estimated cost of such construction was $175,000. As an alternate scheme, I proposed the using of six 6-ft. Chilean mills to be placed between the stamps and tube-mills. My idea was that 4-mesh screens could be used on the batteries, and a duty of 8.5 tons per stamp obtained, followed by classification of the product, feeding oversize to the Chileans, crushing through 16-mesh, and finally, grinding this product in tube-mills after classification. This method was adopted and was in operation 90 days after the decision was reached. The total cost of the reconstruction, including 24 concentrating tables and many minor changes, was $75,000. No additional building was required. At the beginning of operations the ore was amalgamated at the batteries and below the tube-mills. With increasing depth in the mine the baseness of the ore rendered this operation unprofitable, and it was abandoned in September, 1909, in favor of amalgamation of the concentrate prior to cyaniding. The floor-space occupied by the secondary amalgamating tables was used for the Chileans and concentrators and could not have been used for additional stamps and tube mills. Doubtless a natural question here will be why the South African practice of using 4-mesh screens and additional tube-mills was not adopted. This will be answered later.

To summarize: Through the expenditure of $75,000 the capacity of the plant was increased approximately 40%. To have accomplished the same result with stamps and tube-mills would have necessitated the expenditure of $175,000. To have added the requisite number of tube-mills to pulverize the 850 tons of 4-mesh product to 200-mesh would have increased the cost of construction over the Chilean-mill installation, and would have increased the cost of operation decidedly. Had 40 stamps and three tube-mills been put in, the cost of stamping and tube-milling would have remained the same on the increased tonnage; since the power, labor, and supplies would have increased in direct proportion. With the Chilean mills the additional labor for operating was five men. These would have been necessary in either case. For

the Chilean mills 200 hp. is required. Stamps and tube-mills would have required 300 hp. The cost of supplies is approximately 2c. less per ton than for stamping and 3c. per ton for tube-milling. The total cost of pulverizing 80% of the mill-feed through 200-mesh is considerably less with three-stage reduction, as the following figures will show:

Two-Stage Reduction.

	Cents.
Stamping	22.1
Tube-milling	20.6
Total	42.7

Three-Stage Reduction.

	Cents.
Stamping	13.4
Chilean-milling	10.0
Tube-milling	16.6
Total	40.0

I have gone into details for the benefit of the 'doubting Thomases,' many of whom have visited the plant and who have been unwilling to believe that Chilean mills could be operated at a cost of even approximating 10c. per ton milled.

In order to avoid controversy, it may be mentioned here that the above costs per ton will be masked in the yearly figures by the damage to the plant by fire. For three months of the year ended October 31, 1910, the plant operated with 70 stamps and three Chilean mills. However, the above figures are representative of normal conditions. The low cost of operating Chileans is accounted for by several factors: (1) The design and construction of the mills, which were furnished by the Trent Engineering Co. of Reno; they have been most satisfactory in every respect, both as to operation and repairs. Nothing wears out or breaks except the crushing-steel. The horse-power required is 35 each. The capacity in tons pulverized is approximately 75 tons each. (2) The size of the feed to, and discharge from, the mills. Chilean mills fed with 4-mesh and discharging 16-mesh product, work most satisfactorily and produce, even at this mesh, approximately 30% of -200 slime. (3) The character of the ore, which is fairly soft.

The third proposal was to install a sufficient number of tube-mills to handle 850 tons of ¼ in.-mesh battery-product. As above stated, the cost would have been more, since the mill building would have had to be enlarged, and, from the contour of the hill, much grading and filling necessitated. In addition, experience here has not corroborated that of the 'Mines Trials Committee' at Johannesburg, which found that a tube-mill is most efficient when operating on three or four-mesh feed. I do not want to be misunderstood. I believe this to be true on the Rand, where a large percentage of the tube-mill product is leached, and the difference in the ores doubtless accounts for the difference in results. None the less, I have not been able to produce a satisfactory tonnage of -200 slime from one 5 by 22-ft. tube-mill fed with 4-mesh product and operating under conditions similar to the South African recommendations, nor have I been able to secure enough additional

tonnage to compensate the increased horse-power when operating one mill at 32 instead of 27 revolutions per minute.

Conclusions based upon working tests on the Goldfield ore are: (1) 100 stamps, crushing through 4-mesh (0.18-in.) screens, yielding a tonnage of 8.5 tons per stamp, must be followed by ten 5 by 22-ft. tube-mills in order to obtain a -200 product; (2) 100 stamps, operated under above conditions and followed by six 6-ft. Chilean mills, crushing to 16-mesh will require five 5 by 22-ft. tube-mills to deliver a -200 product. This leaves the problem: Can six 6-ft. Chilean mills be operated more economically than five 5 by 22-ft. tube-mills? Operations here make me think so. The following figures may be of interest: In stamping Goldfield ore to 4-mesh, 20% of the discharge passes 200-mesh; of the remaining 80%, the Chilean mills will 'slime' 30%, or 24% of the whole. This leaves 56% to be handled by five tube-mills; all of which has passed a 16-mesh screen, and 80% of which will pass a 30-mesh. Expressed in tons, each mill is fed with 95 tons of this product. Now, by feeding 10 tube-mills direct from the battery with 4-mesh feed, 20% of which will pass a 200-mesh screen, each mill is required to handle 68 tons of 4-mesh feed. Based on figures derived from working tests, the comparative cost is as follows:

Two-Stage Reduction.
100 stamps followed by 10 tube-mills fed with 4-mesh battery -feed:

	Cents.
Stamping	13.4
Tube-milling	30.0
Total (per ton)	43.4

Three-Stage Reduction
100 stamps followed by 5 tube-mills fed with 16-mesh Chilean-mill product:

	Cents.
Stamping	13.4
Chilean-milling	10.6
Tube-milling	16.6
Total (per ton)	40.6

From this the deductions may be made that: (1) Ore may be reduced to 4-mesh in the stamp-battery more economically than to 12-mesh; (2) for the reduction of ore particles to 16-mesh, where 'all slime' is required, stamps, followed by Chilean mills are more efficient than stamps alone; (3) ore may be reduced to -200 mesh in the tube-mills more economically when the mill is fed with 16-mesh than when fed with 4-mesh.

There is one more point to be considered. On referring to the cost of installation, it will be seen that there was a saving of $103,000 in favor of putting in the Chilean mills. Assuming that the future cost of operating these mills will be 15c., an increase of 3c. per ton of ore milled over the cost of operating 140 stamps and 9 tube-mills, and assuming a yearly tonnage of 300,000 tons it will take ten years' operation, or the milling of 300,000 tons of ore, to offset this original saving. It is not believed that the cost will increase to this extent. Rolls and Chilean mills or other methods of comminution were not considered, for the simple reason that the problem was to increase the capacity of a 100-stamp mill,

designed to deliver an 'all-slime' product to the cyanide plant. If this article invites discussion, I shall be glad to go into details more freely at some future time.

Elements of Cost in Operation

Before passing to the detail of operations it may be well to enumerate the factors governing costs and efficiency.

Water.—Water for milling was supplied during the first two years operation entirely by the local water company, coming from the Palmetto range at Lida, nearly 30 miles distant. Recently the mine-water has been conserved and neutralized, and about one-fourth of that consumed is now supplied from the mines. The local company charges at the rate of 50c. per thousand gallons; the total water consumption per ton milled is 220 gallons, or 11c. per ton. This is a lower consumption of water than any wet-crushing mill on record so far as known. This item of cost is included in supplies.

Labor.—With the exception of two or three Slavs employed in roustabout work, the entire mill crew is American. The wage-scale is $3.50 per day of eight hours for ordinary labor; $4 to $4.50 for mill-men; $5 for machinists, electricians and carpenters. Each man makes out his own time on a distribution slip, stating hours worked, department and whether on operation or repairs, which slip, after being checked by the foreman on shift, is delivered to the timekeeper's office, where a daily distribution of labor is made after the following form:

The entire crew, under normal conditions, including all the superintendence, men on current construction, foremen, and others, consists of approximately 90 men. The average daily pay-roll is approximately $400, and the average daily wage, including superintendence, foremen, master mechanic, electricians, etc., $4.44. Labor per ton of ore milled is $0.46; tons milled per man on shift, 9.65.

GOLDFIELD CONSOLIDATED MILLING & TRANSPORTATION CO.

DAILY MILL REPORT

Tons		Assay oz. Au.	Oz. in Heads	Oz. in Tails	Oz. Produced
	Per cent time run				
	Ore received				
	Ore milled				
	Mill residues				
	Conc. plant residues				
	Conc. plant solutions				
	Mill Solutions				
	Oz. amalgam				
	Totals				

CONSOLIDATED MILL

Labor Distribution

Department.	Shifts Operating.	Amount.	Shifts Repairs.	Amount.
Crushing-conveying				
Sampling				
Stamping				
Amalgamation				
Chilean-milling				
Elevating-separating				
Tube-milling				
Concentration				
Neutralizing				
Settling				
Agitation				
Filtering-discharging				
Assaying				
Precipitation				
Refining				
Steam heat				
Surface and plant				
Warehouse-office				
Watchmen				
Salaries				
Total mill labor				
Concentrate plant				
Total labor				

Power.—Power is supplied by the Nevada-California Power Co. at a cost of $6 per horse-power month, based on 90% of the peak load.

The average power load at present is 1500 hp., equivalent to 1.73 hp. or 32c. per ton of ore milled. A segregation of this load for the first year is shown on a chart that will be printed in the continuation of this article. The campaign of construction which has been waged has left no time to devote to bring this up to date.

Supplies.—Supplies, as usual, constitute slightly more than 60% of the total mill-costs. Naturally the distance from bases of supplies, combined with discriminating freight rates, would make this cost higher than is usual for cyanide plants in the United States. In addition to these factors is the exceeding baseness of the ore, which, during the second year's run has shown an increase of nearly two pounds of KCN per ton of ore milled. Had the cyanide consumption remained as low for the second as for the first, the total cost for milling and cyaniding would have been $1.85 per ton instead of $2.12. The following table showing costs for the first three months of this fiscal year and for 1910 may be of interest:

	Nov. 1910.	Dec. 1910.	Jan. 1911.	1910 Average.
Labor	$0.46	$0.476	$0.445	$0.56
Supplies	1.24	1 285	1.263	1.25
Power	0.276	0.325	0.311	0.31
Total	$1.976	$2.086	$2.019	$2.12

In this connection it will not be amiss to note that $40 mill-heads warrant considerably higher costs than does the ore usually sent to a stamp-mill and cyanide plant, and in judging these costs the fact that the ore has averaged approximately $40 for two years, should be taken into consideration.

Operation

Storage and Crushing.—The plant is situated about two and one-third miles from the various working shafts, from which the ore is transported over the company's railroad to the crusher-bins in steel hopper-bottom cars of 50 tons capacity each. Originally a Blake-Dennison automatic inclined weighing machine was used. It was destroyed in the fire and replaced by a Fairbanks-Morse railroad scale, over which the cars pass en route to the crusher. The operation of weighing cars has added nothing to the working cost, and has been decidedly more satisfactory than the automatic machine in this situation, where there are such decided and sudden changes in temperature. The transportation of the ore does not come under the head of mill operations, but I shall give the cost for the benefit of those interested. These costs are based on the actual dry tons milled, which normally will average approximately 25,500 tons per month. The figures include all labor, supplies, and other items incident to transporting the ore, and mine and mill supplies, as well as the maintenance of rolling stock and roadbed.

	Nov. 1910. Cents.	Dec. 1910. Cents.	Jan. 1911. Cents.
Operation	10	6	3
Maintenance	5	5	7
Total	15	11	10

The crusher-bins, as well as the entire crusher-house and belt-way are built of wood and have a storage capacity of 800 tons. A shaker-feeder of the suspended type, driven from the main motor (150 hp.) feeds the primary crusher through rack-and-pinion ore-bin gates. The crusher is a 7½-K Gates. Set for 2½-in. product, it delivers the ore to a 48 by 14-ft. Gates type revolving trommel, with 1½-in. apertures, which delivers the undersize to a 26-in. inclined conveyor (Stephens-Adamson type) and the oversize to two No. 4K short-head crushers, set for 1½-in. product, from which the ore gravitates to above-mentioned 26-in. conveyor. Concave plates of manganese steel for the crushers last approximately seven months and crush 175,000 tons of ore. Manganese steel screen-plates for the trommel lasted 27 months and screened 504,000 tons. The Stephens-Adamson inclined conveyor, set at an angle of 19° 54′ and 369 ft. between centres, delivers the ore to a horizontal conveyor and tripper which distributes the product to a battery storage-bin. This bin is flat-bottomed and has a capacity of 4000 tons. The fire of April 8, 1910, destroyed the inclined

conveyor, so that it is not known what life the belt would have given. The horizontal conveyor has handled 500,000 tons, and will doubtless accomplish that much more before it is discarded. The entire crushing and conveying system is operated for eight hours by three men, two in the crusher-house and one on the conveyors. They handle 850 to 900 tons.

The cost of crushing and conveying is as follows:

Tons	850	850	600
	First 3 months 1911.	Year 1910.	Year 1909.
Labor (cents per ton)	2.1	4.3	3.0
Supplies	0.1	1.3	0.5
Power	1.6	1.5	1.8
Total	3.8	7.1	5.3

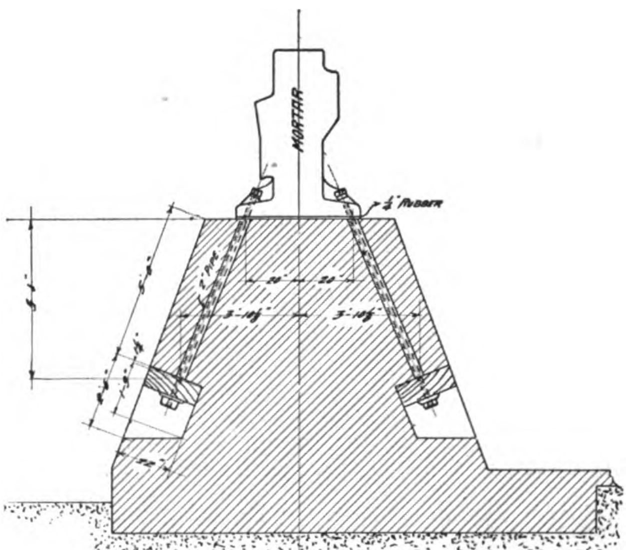
The high cost for 1910 is accounted for by the fire, which destroyed the conveyor system and necessitated the building of an inclined tramway from the battery-bin to crusher-house, and its operation for three months, during which time it was necessary to tram the ore in cars over the battery-bin. The 70 stamps which were not destroyed by fire, were dropping on ore supplied by this tramway in seven days and nine hours from the beginning of the fire.

Stamping.—As stated in the beginning, all details of construction have been published in Bulletin 1438 of the Allis-Chalmers Co., and in this article no attempt will be made to give such details except on new construction. Through rack-and-pinion ore-bin gates, from battery-bin, *via* Challenge feeders, the ore is fed to ten 10-stamp batteries, five right and five left, with 1050-lb. stamps, dropping 108 times per minute through 7 inches. The stamp-weight is distributed as follows: boss, 252; shoe, 185; tappet, 169; stem (3½-in.), 444; total, 1050 lb. The 20 mortars (10,200 lb. weight, 14 in. thick through base) are of the narrow, rapid-discharge, improved Homestake type designed for this company by J. B. Fleming and built by the Allis-Chalmers Co. on his specifications. The cam-shaft for each ten stamps (of hamered iron) is of 6½-in. diameter. During 27 months operation, not one cam-shaft has been broken. The guides originally furnished have been discarded in favor of the 'Couture' guide, designed and patented by A. F. Couture, the master mechanic at the plant. This guide consists of one piece cast-iron (or steel) sole-plate, with tapered sockets for receiving the taper-cored wedges through which the stems operate. The distinguishing feature of the guide is the four-side taper of the sockets and wedges, and the method of dove-tailing which permits of the wedges being reversed when worn. The use of these guides has materially reduced the cost of operation and increased the efficiency of the stamps.

The one-piece mortar block of concrete (mixture 1 : 3 : 3) contains 850 cu. yd. Anchor bolts for mortar and post shoes are

arranged as shown in the following sketch. It may be of interest to note that the concrete withstood the fire of April 8, which destroyed the refinery, store-house, conveyor-way, and twenty stamps. The heat at the mortars was so intense that when the stamps fell, the wrought-iron anchor bolts were drawn out to a fine point. The new equipment was erected on the same foundations with no repairs. There is no sign of serious cracking in the concrete.

One man on each shift feeds the stamps, and one head battery-man on day shift, with a helper, sets all tappets, turns stems, changes shoes and dies, and does all the repair work incident to operating. An amalgamation at the battery has been abandoned, the battery-feeder is alone on his floor, and under such conditions, it seems that 100 stamps is the economic unit for operation. By this



SECTION OF BATTERY FOUNDATION.

it is not meant that total costs for a 140 or 150-stamp mill will not be less, but the actual operation of the stamps will be more, for the reason that while power and supplies will increase in direct proportion to the increased tonnage, labor for 140 or 150 stamps will increase abnormally. Power is supplied for each 20 stamps by one 50-hp. Bullock motor, belted to a countershaft underneath the floor. A rack-and-pinion belt-tightener between the bull-wheel (built-up wood pulley) and this line-shaft controls the power for each 10 stamps.

Chrome-steel cams, bosses, and shoes, and Pennington dies have proved the most satisfactory here. In 27 months only two chrome cams have been replaced. The following comparison of wear of battery-steel may be of interest:

Make.	Days wear.	Price per lb., cents.	Cost per ton ore, cents.
Shoes.			
Chrome (8-in.)	58	5.43	1.88
Pennington (8-in.)	57	5.97	1.91
Midvale (10-in.)	73	6.00	2.00
Dies.			
Pennington (7-in.)	104	5.97	1.06
Chrome (7-in.)	83	5.43	1.25
Midvale (7-in.)	78	6.00	1.30
Tonopah (7-in.)	49	4.00	1.70

The total consumption of steel (shoes and dies) per ton of ore milled in 1909 was 0.98 lb. with a stamp-duty of six tons per stamp through 12-mesh (0.048-in.) screen; in 1910, 0.604 lb.; stamp-duty, 8.46 tons, through 4-mesh (0.18-inch).

The following is a record of time lost and causes thereof for the year 1910, and for three months of the present fiscal year:

Total time lost on account of	1911. Per cent.	1910. Per cent.
Power	0.84	0.70
Water	0.08	1.43
Shoes and dies........................	0.28	0.20
Screens	0.01	0.08
Stems	0.17	0.11
Chilean mills	0.17	0.55
Tube-mills	0.08	0.63
Cyanide plant	0.11	0.21
Miscellaneous	1.40	0.91
Cleaning bat. tank....................	....	0.55
Fire loss	....	7.91
Total.................................	3.14	13.28

A comparison of sizing tests of the 12-mesh (1909) and 4-mesh (1910) product from the stamps is given below, but is of little value owing to the increasing softness of the ore with depth.

Battery discharge through 12-mesh 0.048-in. screen. Duty 6 tons per stamp. Mesh.	Per-cent.	Battery discharge through 4-mesh 0.18-in. screen. Duty 8.46 tons per stamp. Mesh.	Per cent.
Remaining on 20	15.6	10	15.0
" " 40	28.2	30	34.0
" " 60	18.1	50	10.0
" " 80	5.3	80	10.0
" " 100	5.4	100	3.0
" " 200	6.4	150	3.0
Through 200	20.0	on 200	4.0
Through		200	20.0

Originally one 24-in. cone-classifier received the 12-mesh pulp from each battery of five stamps, the spigot product from it gravitating to Dorr Classifiers, the overflow to concentrators. These 20 cones were discarded in favor of two 8-ft. cones, each taking the pulp from 50 stamps. The following is a comparison of the cost of stamping through 12-mesh and 4-mesh screens:

	Table 1. Stamping through 12-mesh 0.048-in. screens. Duty 6 tons.	Table 2. Normal Conditions.		Table 3. Stamping through 4-mesh 0.18-in. screens. Duty 8.46 tons.
	Cents.	Cents.		Cents.
Labor	8.9	6.6	Labor	3.9
Supplies	6.8	7.5	Supplies	4.1
Power	6.4	8.0	Power	5.4
Total	22.1	22.1	Total	13.4

The first and third tables have been taken from the yearly mill-records, and while the various items do not show the proportionate decrease, the total reduction in cost of operating is approximately what would be expected from a 40% increase in tonnage. Several factors besides the coarser crushing have reduced the labor cost in table 1 to the figure in table 3, such as the improved guides, completion of current construction work, etc. Likewise, supplies for 12-mesh stamping in table 1 are not exact, as 200 extra shoes and 200 extra dies, furnished with the plant, did not have to be charged to cost of operation. This is stated merely to avoid confusion, as the sum of the various items in table 1 checks table 2, which is an estimate of the cost of stamping through 12-mesh under normal conditions. The discrepancy between power costs in table 1 and tables 2 and 3 is due to the recent installation of a storage battery which absorbs the mine peaks and virtually puts the mill on a meter reading.

Regrinding.—The second aisle of the mill proper contains the regrinding apparatus, which is divided into two sections, each taking the pulp from 50 stamps. Each section consists of one 7-ft. spitzkasten, three L. C. Trent Chilean mills, one bucket elevator (originally one 54-in. Frenier sand pump), one 8-ft. cone classifier, one 4-ft. spitskasten, three Dorr classifiers, three 5 by 22-ft. tube-mills, and nine No. 3 Deister concentrators. Each section is run by one man on shift and the entire floor is kept in repair by one machinist and one helper on the day shift. This, of course, does not include the re-lining of the tube-mills. The total pulp from the 50 stamps feeding each section, gravitates through wooden launders, lined with 1-in. cast plate (grade 1¾ to 12 in.), to the 7-ft. spitzkasten, overflow from which passes to the main concentrator floor, and spigot product to three 6-ft. Chilean mills. Power for driving these mills is supplied by one 100-hp. Bullock motor belted to an overhead line-shaft. Actual meter readings show that each mill requires 35 hp. to operate when pulverizing 75 tons per day through 30-mesh, and producing 30% of –200 slime from its total feed. When the capacity of the plant was increased to 850 tons, no additional settlers or dewaterers were put in, and for this reason it is impossible to attempt close classification ahead of the Chilean mills, as the additional water required for grinding in them would prohibit economic dewatering in the cyanide plant; hence a great deal of –30-mesh product is sent to the Chilean mills,

but as they have proved to be fairly efficient 'slimers,' the work of the tube-mills is materially reduced, and the apparently poor practice is in reality good. The following is a typical sizing test of the product from the Chilean mills:

Remaining on 10 mesh	Trace
" " 30 "	4
" " 50 "	10
" " 80 "	16
" " 100 "	9
" " 150 "	6
Through 200 "	6
" " 200 "	48
Loss	1
	100

Midvale rolled forged steel has demonstrated its superiority for use in the mills. The dies, 5¼ in. thick, last approximately 125 days; the roller shells, 4 in. thick, average 165 days. Total steel consumed per ton ore milled, 0.32 lb. Screens for a time threatened to become an item of serious expense, as it was impossible to obtain a life of more than six days from the best screens on the market. However, by inserting a strip of 4-mesh screen in the screen frame where the hardest wear fell, thus protecting the screens from unnecessary wear, their life has been doubled.

The cost per ton of operating the Chilean mills, including all repairs and up-keep, is as follows: Labor, 1.8; supplies, 4.1; power, 4.7; total, 10.6 cents.

The pulp from the Chilean mills gravitates to one 18-in. bucket elevator with 28-ft. centres, where it is joined by the tube-mill product, all of which is fed to one 8-ft. cone-classifier, in which partial dewatering is accomplished before passing to the Dorr classifiers. The overflow from this cone passes to the two-compartment 4-ft. spitzkasten and the spigot product from the cone joins the spigot product from the first compartment en route to the Dorr classifiers. The spigot product from the second compartment feeds the nine No. 3 Deister concentrators put on this floor when the Chileans were added. The overflow from this spitzkasten, together with the overflow from the Dorr classifiers gravitates through pipes to the main concentrator floor. Each Dorr machine handles approximately 110 tons of dry ore at a dilution of 3:1, delivering a slime product, 80% of which passes a 200-mesh screen, and a feed to the tube-mills as follows:

	Per cent.
Remaining on 10 mesh	8
" " 30 "	13
" " 50 "	12
" " 80 "	23
" " 100 "	18
" " 150 "	11
" " 200	7
Through 200 "	7
Loss	1
Total	100

The contained moisture is 30%, which is increased to 40% as it enters the mills. The cost of elevating and classifying is as follows, in cents per ton: Labor, 1.2; supplies, 0.5; power, 0.6; total, 2.3. Revere Rubber Co.'s 'Granite' elevator belt and malleable iron bucket lasted 14 months and elevated approximately 400,000 tons dry ore.

Tube-milling.—As stated, each regrinding section contains three 5 by 22-ft. tube-mills (Gates) of the spiral-scoop feed, trunnion-discharge, type. Each battery of three mills is supplied with power from one 200-hp. Bullock motor, belted to the line-shaft. Two of the three mills are belted from this line-shaft and controlled by friction clutches. The third is geared direct from the main shaft, and controlled by cut-off clutch coupling. There is no doubt that individual drives for each mill would be more satisfactory. With the present arrangement the repairing of one clutch hangs up 50 stamps during the operation. Silex lining has been used since the beginning of operations. This lining lasts seven months and is renewed with the following items of expense:

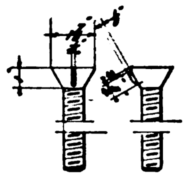
LABOR		
Removing and replacing manhole, removing end-liners	$11.88	
Removing pebbles	3.75	
Removing old lining	11.25	
Re-lining	63.76	
Replacing pebbles	7.50	
Total labor		$98.14
SUPPLIES		
Cast end-liners	$92.48	
Silex, 17,710 lb. at 2.634c. per lb	466.48	
31 sacks cement at $1.10 per cwt	34.10	
Total supplies		$593.06
Total cost		$691.20

This time involved is: Hours re-lining, 68; hours setting cement, 72; total hours lost, 140. While this is less time than is customary at other plants to allow the cement to set, there has never been any trouble on account of starting too soon. The silex consumed per ton of ore amounts to 0.6 lb.; the cost of re-lining per ton of ore milled is 2.3c.; the amount of Danish flint per ton of ore milled is 1.8 pounds.

TOTAL COST OF TUBE-MILLING PER TON MILLED

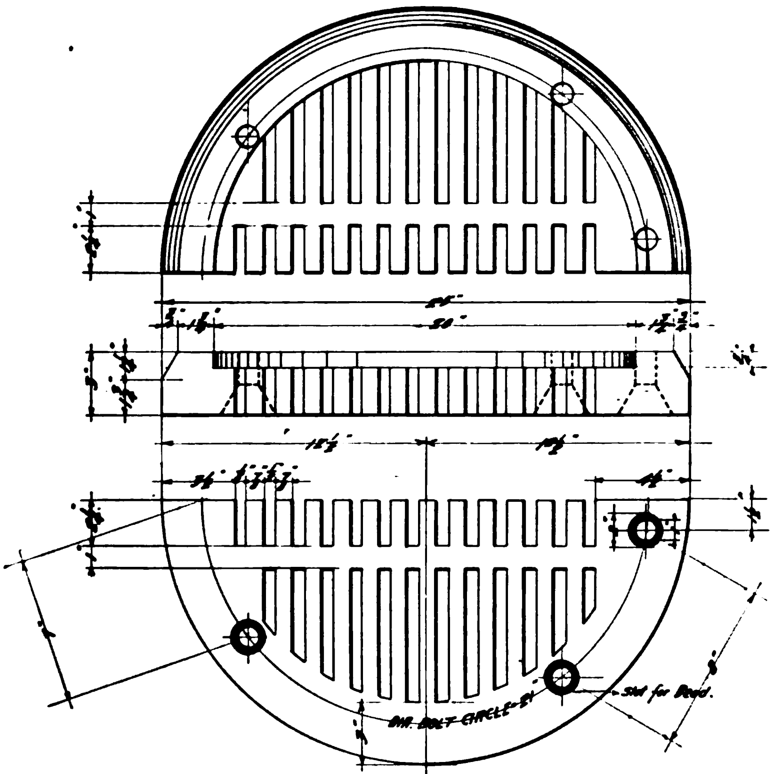
Year	1911	1910	1909
Tonnage	850	850	600
	Cents.	Cents.	Cents.
Labor	1.4	2.1	2.6
Supplies	6.5	6.8	7.0
Power	8.7	9.7	11.0
Total	16.6	18.6	20.6

The design of the discharge screens on the tube-mill has been changed according to the following sketch, and the screens now last six months. The gears and pinions of manganese steel originally furnished with the mills are still in use after 27 months operation and will probably wear 12 months longer.



The product from the tube-mills is graded as follows:

		Per cent.
Plus	80 mesh	5.8
	100 "	13.2
	150 "	11.6
	200 "	17.8
Minus	200 "	50.6
	Loss	11.0



CAST-IRON SCREEN FOR DISCHARGE END OF TUBE-MILLS

Although the ore is classed as 'soft' ore, 40% of the product fed to the tube-mills is extremely hard quartz and is reduced to minus 200 product with difficulty. This was very noticeable when testing the mills with 4-mesh product from the batteries. The

pulp discharged from the mills contains large quantities of coarse, rounded particles, which so accumulate after a few hours run that it becomes necessary to shut off the feed and grind the mills out.

After classification, the final product, which, as can be seen from the flow-sheet, is produced from two sources, the Dorr classifiers and spitzkasten, shows the following analysis:

	Per cent
Plus 80 mesh	0.4
100 "	2.2
150 "	8.1
200 "	9.2
Minus 200 "	79.1
Loss	1.0
Concentration	100.0

Concentration

The third aisle of the mill is the main concentrator floor, containing 30 8-ft. Callow tanks, and 60 primary and 16 secondary concentrators (all No. 3 Deister slimers). The concentrators were put in after a competitive test on this ore with the suspended vanner; the Deister No. 3 machine winning by a wide margin. Each machine has the capacity, when followed by secondary concentrators in the ratio of 1:5. of handling 11 tons of dry slime, 80% of which will pass a 200-mesh screen and all of which will pass a 100-mesh, and each will concentrate therefrom 72% of the gold in the ore into 1000 lb. of concentrate. From this, 20% of the gold is recovered by amalgamation, leaving 52% to be recovered by further treatment of the concentrate. When the facts are taken into consideration, that the ore has been slimed before any of the concentrate has been removed, that 80% of the material recovered is –200-mesh, and that the ore can not be classed as a concentrating ore, the performance seems remarkable. Repeated monthly tests on a general sample of the tailing from the cyanide plant fails to show any appreciable gold recoverable by concentration. When the capacity of the plant was increased, 18 additional tables were placed on the regrinding floor, and 6 on the secondary floor. As stated, the tables on the regrinding floor take their feed from the spitzkasten, this feed assays 20% higher than the feed on the main floor. For this reason, the middling from the 18 upper tables is re-concentrated on the main floor and the tailing from the upper floor mixed with middling from the 60 primary tables.

Although 30 Callow tanks were placed on the main floor and were used for dewatering when the plant crushed 600 tons, at the present only 16 are in commission, since less water is now used for crushing. The Deister machine operates more satisfactorily on a pulp containing 3 to 3½ parts water. They are driven at 300 strokes per minute through $^{7}/_{16}$ in. and require 0.73 hp. each to operate. One man per shift operates the 60 primary and 16 secondary tables; one man with a helper keeps all the machines in repair. This work includes all mechanical up-keep, as well as cleaning

Product	Dilution	Width of launder	Height of launder	Grade of launder	Dry tons handled in 24 hr.	Conveying pulp.
		In.	In.	In. per ft.	Tons.	
4-mesh from batteries	4:1	8	8	1¾	255	From 30 stamps to spitzkasten.
12-mesh from batteries	6.5:1	8	8	⅝	200	(1909) from 30 stamps to 8-ft. cone.
−4 mesh to Chileans	3.3:1	8	8	1⅛	400	From spitzkasten to three 6-ft. Chilean mills.
−30 mesh from Chileans	3.3:1	8	8	⅝	400	From three 6-ft. Chilean mills to boot of B. & B. elevator.
Tube-mill discharge, 50% −200 mesh	1.5:1	6	5	1¾	100	From one 5 by 22-ft. tube-mill to boot of B. & B. elevator.
Mixture of tube-mill and Chilean products	2.6:1	8	8	¾	700	From three 5 by 22-ft. tube-mills and three 6-ft. Chilean mills to 8-ft. cone.
Mixture of tube-mill and Chilean products	2:1	6	5	1⅛	200	Feed to one Dorr classifier.
Final product from tube-mills	3:1	5½	8	⅜	330	Feed to 30 No. 3 Deister slimers.
Product from Callow tanks	3:1	3½	3½	⅞	22	Feed to 2 No. 3 Deister slimers.
Concentrate	9:1	3	3	1 1/16	3.5	From 6 No. 3 Deister slimers to main launder
Middling	3:1	3	3	¾	10	From 6 No. 3 Deister slimers to main launder.
Tailing	5:1	5	5	¾	52½	From 6 No. 3 Deister slimers to main launder.
Concentrate	9:1	4	10	¾	50	Main launder to concentrating plant.
Tailing	5:1	10½	10½	5/16	800	From 100 stamps.
Clear water		18	14	⅛	4000	Clear water overflow from 800 tons.

floors, machines, and pulp-thickeners. The helper brushes the mineral edge of each table every morning with a dilute HC1 to remove precipitated salts. Each deck is brushed thoroughly every 60 days.

The concentrate from all floors, approximately 5½% by weight of the ore milled, gravitates to the concentrate-treatment plant and will be discussed later. Middling from the primary tables, amounting to 15% by weight, is re-concentrated without further regrinding on the 16 secondary tables. Reference to the flow-sheet will show the disposition made of each product. The cost for all renewals for 94 tables, including head motions, deck, riffles, etc., has averaged $150 per month for twelve months, or at the rate of $1.60 per table per month, or 0.06c. per ton milled. The labor for repairs and maintenance has been approximately $3 per table per month, or 1.2c. per ton milled, which includes everything incident to keeping the tables in first-class operating condition. The following table shows the total cost of concentrating, including the above items of repair:

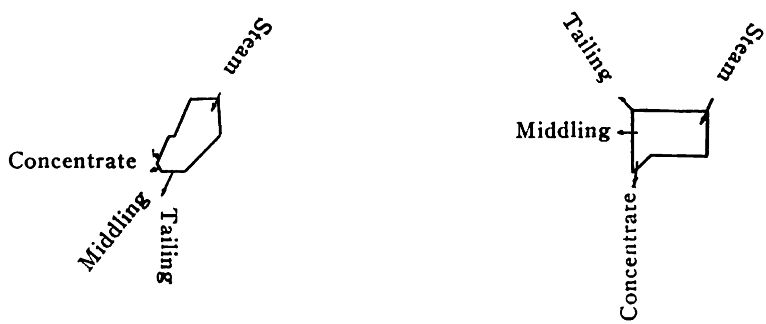
Year	1911	1910	1909
Tonnage	850	850	600
	Cents.	Cents.	Cents.
Labor	3.5	4.0	4.0
Supplies	0.5	0.4	0.4
Power	1.5	1.5	1.8
Total	5.5	5.9	6.2

As pulps are being conveyed here with more variety in size of solids and degree of dilution than in the gold mills of this country or elsewhere, it is hoped the tables on page 562 will prove of interest:

Neutralizing and Dewatering

The lime-mixing plant is situated about 200 ft. from the mill proper and consists of storage bins (capacity 120 tons) into which the lime from the railroad cars is shoveled. Lime from these bins is slacked and dumped into two 4-ft. Wheeler pans, from which the mullers have been removed. These pans act simply as stirrers and deliver a continuous stream of milk of lime, which is laundered to the mill, with branch launders to the dewatering tanks, Pachuca agitators, and concentrate-treatment plant. The bulk of the lime consumed is added to the main launder conveying the table tailing to the dewaterers. No lime is added at the battery, and for this reason there is not the serious trouble with concentrator decks noticeable at most cyanide plants. The lime for neutralizing is so regulated that the overflow water from the dewaterers titrates 0.4 to 0.5 lb. CaO per ton of water. This mill-water gravitates through two clarifying tanks to mill-water sump-tank, from which it is elevated by means of two 10 by 12 Aldrich pumps to the mill-water supply tank behind the stamps. The water leaving the batteries is a trace acid to phenol, with acidity has increased to 0.2 when the

pulp reaches the concentrators. The ferrous salts generated during crushing are neutralized and oxidized en route to the dewaterers. The 16 dewatering tanks, 29 ft. 6 in. by 12 ft., with 16° false cones, are arranged in two aisles of 8 tanks with 7 ft. difference in elevation. The total capacity of the dewaterers is 96,000 cu. ft., equivalent to 120 cu. ft. per ton of ore. or 20 cu. ft. per ton of pulp. Each tank is equipped with a central well for the inflow of pulp, and



FLOW OF MATERIAL OVER DEISTER TABLES

peripheral launders and pipe decanters for handling the clear water. The helper in the cyanide plant regulates the lime and settlers, decants and transfers the charges. The cost of these operations is as follows:

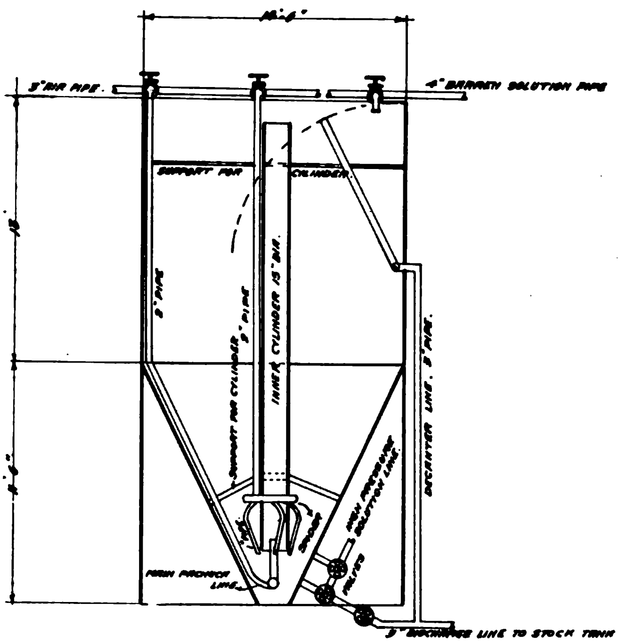
	Neutralizing, cents.	Dewatering, cents.
Labor	0.6	1.4
Supplies	4.1	4.1
Power	0.1	0.1
Total	4.8	5.6

Cyanidation

From the dewaterers the pulp, containing 40% water, is pumped to a battery of ten 15 by 45-ft. Pachuca agitators, arranged in series, and is diluted to 1½:1 (Sp. gr., 1.35) with soltuion from the wash-solution tank (No. 3 above) in the Butters filter system. The solution is increased to 1.2 lb. KCN per ton and the alkalinity 0.5 lb. in terms of CaO and maintained at these strengths for 26 hours. Lead acetate to the acount of 1/6 lb. is added at the beginning of treatment and again in tanks 4 and 8 of the series. This seems unusual for gold ores, but the decomposition of the complex sulphides in the Goldfield Consolidated Mines Co.'s ore makes it necessary. All attempts to reduce the amount have resulted disastrously. Various oxidizing agents, together with bromo-cyanide, have been tested on a working scale without commercial success. The bromo-cyanide gave increased extraction, but the cost was prohibitive, except on high-grade ore. As stated, concentration

tests on the tailing are made on a general monthly sample, but the loss is not in recoverable concentrate.

There is one feature which is most interesting. The assay value of the –200 product in the tailing is considerably higher, nearly 30%, in fact, than the product remaining on 200 and 150-mesh screens. This is true in both the mill and concentrate plant and is doubtless due to the presence of very brittle and insoluble alloys of gold. A picked sample of the higher grade ore supplied to the mill gave by analysis the following percentages of elements which tend to confirm this theory: Bismuth, 0.25; antimony, 0.055; tellurium, 0.025; selenium, 0.050.



PACHUCA AGITATOR FOR CONCENTRATE TREATMENT

In the agitators approximately 80% of the value of the ore after concentrating is dissolved. Previous to connecting the tanks in the series they were operated as 4 batteries, two of 3 and two of 2 tanks. At a gravity of 1.35, with a loss of 4 hours for transferring, the average period of agitation was 23 hours. After the publication of M. H. Kurylas' article on the results of continuous agitation at Esperanza, I wrote the management of that property asking for additional information, which was given. Using Mr. Kurylas' sketch, the Pachuca tanks were connected in series by means of 8-in. pipe connections, and experiments begun. These were not favorable to continuous agitation, and for six months thereafter the tanks were worked intermittently. The failure of the system was not explained until recently. After a new compressor with much more capacity than those previously

used for agitating was put in, the tanks were again connected in series and with marked success. The only reason known here for the failure at first is that uniform agitation throughout the series is most essential. With the inadequate amount of air at that time, this could not be maintained, and the tailing, gravitating from the series to the filters, had not received uniform treatment. The following assays will give an idea of the first and second trials.

	First Trial.			*Present Operation.*		
		Oz. Au.			Oz. Au.	
	Gravity.	Pulp.	Sol.	Gravity.	Pulp.	Sol.
Heads	....	0.44	....	....	0.62	....
Agitator No. 2.......	1.36	0.17	0.17	1.35	0.31	0.21
" No. 4.......	1.36	0.16	0.17	1.34	0.26	0.256
" No. 6.......	1.34	0.16	0.18	1.33½	0.20	0.30
" No. 8.......	1.33	0.14	0.193	1.33	0.14	0.328
" No. 10.......	1.33	0.12	0.20	1.32	0.10	0.348
Filter tailing	....	0.10	....	....	0.08½	....

The lighter gravity in the last tanks of the series is accounted for by the addition of solution for dissolving chemicals. The screen tests on the various tanks is as uniform as could be expected. There are so many different kinds of ore being fed to the mill, that it is impossible to make a definite comparison of the two methods of treatment, as the ore varies within wide limits during a week's run. The continuous system during the first run was alternated weekly with the intermittent three times, with an apparent increase in tailing loss of 30 cents. The present operation shows a decrease of nearly the same amount, but on a higher grade of ore. The following analysis of the solution before precipitation shows some of the difficulties encountered in the treatment:

Analysis of Cyanide Solution at G. C. M. Co. by Von Schultz & Low.

(The percentages refer to the weight of the solution. Total solids by evaporation, 0.3932%. Sp. gr. solution at 70°F., 1.002.)

	Per cent.
Silica	0.00198
Ferric oxide	0.00014
Alumina	0.00020
Manganese	Trace
Lead	Trace (faint)
Bismuth	Trace (faint)
Cadmium	Trace (doubtful)
Copper	0.03686
Arsenic	0.00010
Antimony	0.00014
Nickel	0.00010
Cobalt	0.00020
Zinc	0.00330
Selenium	None
Calcium oxide	0.02837
Magnesium oxide	0.00014
Phosphorus pentoxide	Trace (faint)
Sulphur trioxide	0.03282

	Per cent.	
Chlorine	0.03260	
Tellurium	0.00008	
Gold	0.185	(oz.)
Silver	0.068	(oz.)

The total sulphur was determined as sulphur trioxide, and no attempt was made to determine the form of combination of the elements found.

The cost of cyaniding, which, in the method of accounting employed at the plant, is really the cost of agitating, includes all chemicals necessary for the dissolution of gold, and all labor connected with the operation of tanks, pumps, and compressors, is given below. Naturally the cost has varied considerably during the three years' operation, due to the increasing cyanide consumption. The mechanical cost is practically constant. Approximately 75 cu. ft. of free air per minute is required for each tank of 85 dry tons, equivalent to 6 hp. per tank. Power per ton agitated is 2c. Maintenance and repairs for tanks, compressors, pumps, and pipe-lines is less than one cent per ton agitated. The total cost, including all the above items of chemicals and repairs, is as follows:

Year	1911	1910	1909
Tons	850	850	600
	Cents.	Cents.	Cents.
Labor	2.8	2.6	2.8
Supplies	56.6	51.0	40.3
Power	2.4	2.5	3.2
Total	61.8	56.1	46.3

It may be well to state here, in order to correct erroneous impressions given in articles on cyanidation of the low-surface ores at Goldfield in mills other than those of the Goldfield Consolidated Mines Co., that it is a characteristic of the sulphide ores of the district to increase in refractory elements with the increase in value. One notable shipment of high-grade ore contained nearly 2.5% tellurium with a gold content approximating 2%. With depth the baseness of the ore naturally increases. It has been demonstrated here to the satisfaction of all interested, that the lower-grade ore is more amenable to treatment by cyaniding than the higher-grade ores, and that the percentage of gold extracted does not necessarily increase with the increase in value. This is the inference made in some recent articles commenting on the extraction of one of the local mills, operating on $12 upper-level ore. It is stated that this mill makes a saving of 93% on such ore, which approximates the saving made at the Consolidated mill on $30 ore. It is interesting to note that immediately after the fire of 1910, when the higher-grade ore from the deeper levels was shipped to smelters, and the mill heads lowered to something like twice the value of those at the mill referred to, the extraction at the Goldfield Consolidated mill averaged 96%. As soon as normal operations were resumed with the mill treating all the deep-level ore, with consequently increased value in the feed, the percentage recovered dropped to 94½, which

was the average for the year of 1910. Attention may be called to the performance of the old Combination mill, which for crudeness of design and lack of conveniences rivaled the plants in question. In this mill, which treated the upper-level ores, and consequently those least refractory, an average extraction of 95% was maintained. It has been the policy of the Consolidated company to include in the cost in its monthly and annual reports the residues from the concentrate treatment plant which could not be shipped at a profit, with the tailing losses at the mill. As a consequence the published report of recoveries has not been a statement of the metallurgical efficiency, but more nearly the statement of the percentage of value applicable to expenses and profits. Therefore, the greater metallurgical efficiency in the concentrate plant resulted in a lower reported extraction in the mill.

Filtering

From tank No. 10 of the agitator series the pulp gravitates to two 34 by 12-ft. pulp-storage tanks fitted with mechanical stirrers. These tanks are situated about five feet above the top of the filter-boxes. On the same level are two -wash-solution tanks which will be called 'No. 3 and No. 4 above' in the following description. Reference to the flow-sheet will assist in following the cycle of operations. From the pulp-storage tanks through a 16-in. pipe-line the pulp gravitates to two steel filter-boxes with 6 hoppers each. The boxes contain 168 leaves each and are worked as one unit. All valves on the 16-in. filling and emptying line, are actuated hydraulically with levers from a central switchboard. After the cake has been formed the excess pulp gravitates through a 16-in. line to an excess-pulp tank from which it is elevated to the two filling tanks by means of 5-in. Morris pumps. Wash solution is run in from 'No. 4 above,' which tank is filled with precipitated solution direct from the barren sump-tanks. After washing and dropping the cake, the supernatant solution is decanted to the excess-solution tank from which it is elevated to 'No. 3 above' by means of 4-in. Krogh pumps. The solution for diluting the charges from the dewatering tanks is drawn from 'No. 3 above' and discharged into No. 1 Pachuca of the continuous system. In this way, by using the solution which has been in contact with the filter-cake once, for agitating the new charges, the accumulation of gold in wash solution, which is the main 'talking point' of the opponents of submerged vacuum-filtration, is avoided. The value in wash solution here is directly dependent on the efficiency of the precipitation, and while absolute recovery of dissolved metal is not claimed, the loss is unavoidable. Furthermore, it has been demonstrated beyond a doubt that additional extraction is caused by the leaching action of the wash solution through the cake, which cannot be accomplished by means of agitation unless the tailing from the agitator is treated with a freshly precipitated solution. The amount of gold dissolved on the filter may be *reduced* by longer agitation, but the point has never been reached where the wash solutions do not dissolve gold in appreciable quantities. Ordinarily, this amounts

to 30c. per ton. During June of 1910, when the plant was running at 70% capacity on account of the loss of 30 stamps by fire, the pulp was agitated for six hours longer than had been customary with full tonnage. The tailing from the Pachucas assayed lower, but the filters still gave additional extraction, though to a less degree than was normally the case, indicating that all the gold dissolved by longer agitation would have been recovered on the filter. The following comparisons, taken from the June records, is interesting:

	Oz. Au.
Tailing samples of 91 charges from Pachucas to filters (thoroughly washed with water)	0.095
Samples of 200 Butters filter discharges from same pulp	0.088
Extraction on filter	0.007

It is also believed here that the completeness of the displacement of dissolved gold in the cake, as well as the additional dissolution of gold on the filter, is not as dependent on the quantity of wash solution passed as on the length of contact. Consequently, I prefer to wash for a longer period with a reduced vacuum. The time allowed for this operation ranges from 85 to 100 minutes. During this time 1⅛ tons of wash solution per ton of ore is passed through the cake, of which solution the following are typical assays, the moisture in discharged pulp being 33⅓ per cent:

	No. 1. Oz. Au.	No. 2. Oz. Au.
Effluent solution making cake	0.192	0.20
Effluent wash at end of 10 min.	0.17	0.19
" " 20 "	0.166	0.185
" " 30 "	0.164	0.182
" " 40 "	0.158	0.168
" " 50 "	0.08	0.092
" " 60 "	0.04	0.03
" " 70 "	0.03	0.025
" " 80 "	0.025	0.022
" " 90 "	0.015	0.015
" " 100 "	0.006	0.015
Sample solution in hoppers 10 min.	0.001	0.008
" " 20 "	0.001	0.008
" " 40 "	0.001	0.007
" " 60 "	0.002	0.007
" " 80 "	0.002	0.010
" " 100 "	0.002	0.010

Assuming that the amount of solution discharged during the 10-minute intervals is constant, which is perfectly fair, the average value from the assays of the effluent wash for test No. 1 is 0.085 oz. gold, and for test No. 2, 0.092 oz. gold. Naturally this average value is not correct, since the solution is sampled intermittently during a process of gradual reduction in grade, but for the purpose in view it is sufficiently close. It will be noticed from the tables that only between the 40, 50, and 60-minute samples is there a decided drop in value. At the end of the fiftieth minute, in both cases given and in numerous tests made, 87% of the total value of the effluent wash has been picked up. The last fifty minutes

and 50% of the total solution passed, are required to accumulate 13% of the total value. In both cases, based on the original assumption that the flow is constant for each 10-minute interval, only 1c. per ton of ore is removed from the pulp during the last ten minutes of washing. In test No. 1, assuming that the total moisture of $33\frac{1}{3}\%$ is of the same value as the one-hundredth-minute sample, which, on the face of it, is unfair to the filter, the unwashed gold left in the pulp amounts to 0.003 oz., or 6c. per ton. In test No. 2, in which the wash solution is high, due to the higher value in the barren sumps, the unwashed value is 0.0075 oz. gold, or 15c. per ton. By referring to the subject of precipitation, it will be noticed that during the whole time of operation the barren sump solutions have averaged 0.004 oz. gold; from which fact it is safe to state that the soluble gold left in the discharge pulp has not exceeded 6c. per ton, discharged. When the filter is credited with the additional extraction obtained on it, which cannot be obtained with those filters passing large volumes of wash solution in a short period of time, the adoption of the submerged vacuum-filter has certainly been justified on this ore. The filter plant contains 33,600 sq. ft. of filter surface and averages 50 lb. of slime filtered and 120 lb. of solution recovered per square foot of filter per day. The usual Butters automatic acid-washing apparatus is used to circulate a $\frac{1}{2}\%$ solution of HCl through the leaves to remove the carbonate of lime. Ten leaves are acid-treated each day, which makes a complete cycle in 34 days. It is worthy of special mention that the original filter-cloths are still intact after 27 months operation, and that they will undoubtedly wear 12 months longer. The tailing is discharged through twelve 12 by 12-in. flat Wheeler gate valves at the bottom of the hoppers into a tunnel, through which it gravitates to the slime pond. For the information of a certain distinguished gentleman who writes an annual review of cyanidation, and for the attention of his readers who may be misinformed, I beg to state that never, since the beginning of this operation, has wash-water been added to the tailing discharged from this plant to be settled and returned to the mill for subsequent precipitation, and that the percentage of moisture in the discharged pulp does not now exceed and never has exceeded 35%. Any excess solution built up from the incoming moisture from the de-waterers is carefully precipitated, stored until the assay-value has been determined, and wasted only when the value does not warrant further expense. The following data are representative of normal operations:

Filter Cycle	Min.		Min.
Filling boxes with pulp	10	10	10
Making cake	60	"	80
Emptying excess pulp	14	"	14
Filling with wash solution	10	"	10
Washing	85	"	100
Decanting and discharging	20	"	20
Total time per cycle	3 hr. 15 m.	to	3 hr. 50 m.
Tons per cycle	125 to 150		

Cost of Filtering, Including Up-Keep of Slime-Pond

Year	1911	1910	1909
Tons	850	850	600
	Cents.	Cents.	Cents.
Labor	3.6	4.1	4.3
Supplies	1.2	1.7	1.9
Power	2.2	2.6	3.1
Total	7.0	8.4	9.1

Precipitation

The solution from the Butters filter is clarified in three 36 by 36-in. 60-frame Perrin presses, which are fed by one 4-in. Morris centrifugal pump direct-connected to a motor. They have the capacity to clarify 2000 tons of filtered solution per day, equivalent to 1300 lb. of solution per square foot of filter-surface. To this operation must be accredited part of the success obtained here with zinc dust precipitation. At times it has been necessary to pass part of the solution to the precipitation-tanks without clarification, and each time it has resulted in a precipitated solution of high metal content. The high-grade solution from the concentrate plant is clarified in a similar press and precipitated in the mill presses. The solution from the clarifying-presses gravitates to three 28 by 8-ft. redwood tanks, two of which are for the mill and one for the concentrate-plant solution. The usual Merrill equipment is used for feeding and emulsifying the zinc dust, which gravitates through 1-in. rubber hose to the suction pipes of two 7 by 9-in. Aldrich pumps, which deliver the solution and zinc dust to four 30-frame, 48-in. triangular Merrill precipitation-presses. These presses have 1680 sq. ft. of filter-surface and 140 cu. ft. of storage for precipitate, equivalent here to 1.6 tons solution per square foot of filter per day, and to ¼ cu. ft. of storage per pound of daily precipitate. Each press, when filled at a pressure not exceeding 5 lb. per square inch, will hold approximately 2500 lb. of precipitate containing 30% H_2O.

In figuring precipitating equipment for silver ores, the filter surface is not the only consideration, as can be seen from the following instance. In evolving a process for treating a high-grade silver-gold ore, it was decided to precipitate 600 tons of solution from 120 tons of ore. The equipment, which a zinc-dust process company agreed to furnish, contained 384 sq. ft. of filter-surface, which would have been satisfactory for filtering. The storage-room in this equipment amounted to 32 cu. ft. The amount of daily precipitate at this plant is estimated at 400 lb. It can readily be seen that had this equipment been put in it would have been necessary to clean the presses every fifth day. It was finally decided to put in 800 sq .ft. of filter surface and 70 cu. ft. of storage, and clean up three times per month. It seems that it would be advisable for the patentees of precipitating-presses to design a special press for silver ores with a greater proportion of storage

room to filter-surface than the standard press in use has. It has been found necessary here not to allow the pressure to exceed 5 lb. per square inch; at a higher pressure the precipitate has a tendency to cake, which prevents the inflowing solution from filtering through the excess zinc dust and being precipitated. This experience is in direct contradiction to the theory that precipitation is complete in the pipe-line, but it has not been possible here to corroborate this. After cleaning the presses, in order to insure satisfactory precipitation, it is necessary to use $^{3}/_{5}$ lb. of zinc dust with the first 250 tons pumped through the presses. After the filter-cloths have been coated with this excess zinc, the amount used varies from $1/8$ to $^{1}/_{5}$ lb. per ton of solution. During the first 18 months' operation the plant was supplied with most inferior zinc dust, which resulted in a high consumption of this chemical, as can be seen by referring to the table on page 16. In the latter part of 1910 it was decided to have this commodity shipped in metal-lined cases, similar to the cyanide package. The beneficial effect was immediate, and although the first cost is $1/4$c. per pound higher than the quotations for barrel packages, the consumption of zinc dust is reduced enough to offset this expense many times.

At the beginning of operations the strong and weak solutions were precipitated in separate presses. During this time the total zinc in the precipitate averaged over 30%. It was decided later to pump the strong solution, which titrates 4.5 lb. KCN and averages $1\frac{1}{2}$ oz. gold per ton, through the weak-solution presses. By doing this the zinc has been reduced to 15%, and no more zinc dust is required to precipitate the $1\frac{1}{2}$-oz. solution than is used on the mill-solution. This is one reason why it is believed here that the condition of the press is more important for efficient precipitation than length of pipe-line. Since the metal-lined zinc dust package has been used, three presses are kept in constant operation, and the fourth cut-in only at the beginning of the bi-weekly clean-up. The clean-up car, which is steam-jacketed, runs on rails underneath the presses, as shown on page 14. This change was made in order to avoid handling or transferring the precipitate, and is very satisfactory. Six hours are required for two men to clean a press and have it ready for operation. The work of cleaning is done by the refinery crew, which consists of three shifts of two men each. All filter-cloths for the clarifying and precipitating-presses are cut by the filter operators who have some little time during each cycle for this work. A double thickness of twill is used for filtering; when the outside cloth becomes worn, it is taken off, burned, and added to the precipitate.

The efficiency of this method of precipitation is shown in the following table:

	1911.	1910.	1909.
Value of Mill Solution—	Oz. Au.	Oz. Au.	Oz. Au.
Before precipitation	0.200	0.200	0.230
After precipitation	0.003	0.004	0.005
Percentage of recovery	98.5	98.0	97.8

Value of Concentrate Plant Solution—	1911. Oz. Au.	1910 Oz. Au.	1909 Oz. Au.
Before precipitation	1.56	1.22	1.40
After precipitation	0.014	0.016	0.017
Percentage of recovery	99.0	98.7	98.8

In the cost of precipitation is included the power required to elevate the solution to the press-room and the total cost of operating the clarifying-presses. The latter may seem unjust to the zinc dust precipitation, but since it is a detail absolutely essential to the successful operation of this method, it should be charged against it. The solution from the filters, although apparently clear, contains a minute amount of floculent slime which would cause the pressure in the precipitating-presses to rise if it were allowed to pass to them. In addition, the unclarified solution would precipitate, from a physical standpoint, on zinc shaving, and as stated, it is thought here this cost is properly charegd against the precipitation. The following table shows the cost, including cleaning-up and items mentioned above:

	1911	1910	1909
Tonnage	850	850	600
	Cents.	Cents.	Cents.
Labor	1.0	0.9	1.3
Supplies	4.9	8.7	9.0
Power	1.4	1.4	1.7
Total	7.3	11.0	12.0

In spite of the fact that the heat was so intense it warped 12-in. I-beams, destroyed part of the steel structure of the press building, and burned the zinc in the presses, they passed through the refinery fire without damage, yielded 11,000 oz. of gold which they contained, without loss, and were in operation four days after the disaster.

Melting

The melting-room, as originally designed, contained a double-muffle drying-furnace and four Faber du Faur tilting furnaces for treating this precipitate. Apparently not enough experimental work was done on this part of the treatment while engaged in making the design for the plant. The process as outlined consisted of nitre roasting the precipitate, with subsequent melting in the tilting furnaces. The bullion from this method averaged about 250 fine in gold and silver. The analysis given will explain why. Later, acid-treating tanks were put in, which materially reduced the amount of precipitate to be melted, and by the addition of pyritic concentrate to the flux, enough copper and lead were converted to matte to raise the grade of the bullion to 425 fine, gold and silver. The following is a typical analysis of the bullion originally made at the plant:

	Per cent.
Au	34.87
Ag	5.75
Cu	40.50
Pb	12.95
Zn	3.48
Cd	1.31
Fe	0.18
Mn	Trace
As	0.10
Sb	0.15
Bi	0.07
Ni	0.16
Co	0.03
Te	0.39
S	0.10

The process was decidedly unsatisfactory. The fire, which originated in this melting-room, through the failure of a defective bushing in the fuel-line, made it imperative to go ahead with the work which had long been planned but postponed on account of other work which seemed more necessary. Much work was done with a view to finding some wet method which could be economically applied to the precipitate. The large amount of base metals made the cost of reagents prohibitive. Electrolytic parting of the base bullion would have made it necessary to carry a large stock of silver in order to secure the correct proportion of silver to gold for rapid parting. This idea was abandoned on account of the high cost. Cupellation of the briquetted precipitate, as practised at the Homestake, was impossible on account of the high percentage of copper. Laboratory work with a modification of the Tavener process, in which enough sulphur obtained from concentrate was added to matte the copper, yielded a clean lead bullion, and low-grade slag and matte. The cupellation of this base bullion left a gold bullion carrying approximately 580 parts gold and 80 parts silver.

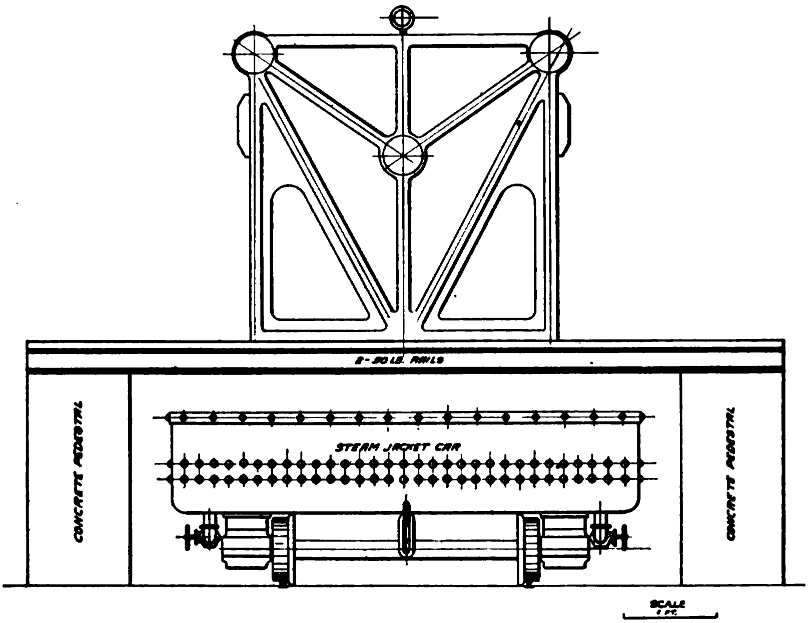
About the time it had been decided to briquette the precipitate with litharge and concentrate and smelt in a small reverberatory with suitable flux, Henry Hansen, the mill superintendent of the Pittsburg Silver Peak Co., at Blair, Nevada, told me of some experiments which he had been making with a small blast-furnace for smelting his briquetted precipitate. The refining process at that mill was at that time similar to Homestake practice. He had found the smelting of the briquetted material in the cupels a tedious operation, and had made some tests in a small blast-furnace which he was using for cleaning his cupel slags and for other clean-up work. The bullion from this furnace was cupelled, and the total time and expense of melting were reduced materially.

Smelting precipitate containing 40% copper on a lead basis did not sound very attractive as blast-furnace work at first, but after experimenting here it was decided it could be done very efficiently and economically. The precipitate is now treated in the following manner: The upper floor of the refinery is connected with the pre-

cipitating-room by a 30-in. gauge track with branches to each of the four presses. The steam-jacketed clean-up car is run underneath the press to be cleaned, the precipitate is dropped into it and hoisted to the refinery, where the total weight is determined on platform scales. After deducting the weight of the car and the moisture, fluxes are added in the following proportion:

Dry precipitate	100
Litharge	100 to 125
Blanket concentrate (S 35%, SiO^2 30%)	60 to 75

Flue-dust and floor sweepings in quantities to reduce the moisture to approximately 9 per cent.



MERRILL PRESS AND CLEAN-UP CAR

The admixture of precipitate and fluxes in the car passes to a hydraulic elevator which delivers it at the feed-hopper of a two-mould Boyd press. This press was purchased from the Selby Smelting & Lead Co. and has a rated capacity of 30 tons in 24 hours. The dies have been changed to make a circular brick 4½ in. diam. and 3 in. thick, since the rectangular one was too large and unwieldly for use. Two men have no difficulty in fluxing and briquetting 2000 lb. of precipitate in 8 hours. One man feeds the press and the other receives the briquettes on rectangular trays, which are stacked in jacketed drying-carts and allowed to dry for 48 hours. Steam was used for drying during the winter when the heating plant was in operation. Having an excess of air, it was decided to heat it in one of the muffles of the drying-furnace, and

allow the heated air to circulate through the trays of briquettes. This arrangement is very satisfactory and is more economical than the use of steam jackets.

The coke-bin is situated on this floor of the refinery and the coke can be shoveled to the furnaces with no extra handling. The blast-furnaces are of the cylindrical type, 20 inches in diameter at the tuyere line with riveted steel jackets and provided with a removable curb, mounted on wheels. The jackets are arranged to be suspended from 15-in. I-beams which pass through the concrete retaining-wall. A Connersville blower direct-connected to a 10-hp. motor supplies a blast of 3 oz. per square inch for both blast-furnaces and the double English cupelling-furnace. This blower has a capacity of 10 cu. ft. of air per revolution and is operated at present at 250 r.p.m. Arrangements are being made to reduce the speed, as the supply is in excess of the amount required. The gases from all furnaces pass to a dust-chamber which contains 2700 cu. ft. There are nine take-out hoppers inside the building through which the flue-dust is drawn after each melt. The product recovered from the flue never exceeds 400 lb., and contains less than $500 in gold from a total value in the melt of $400,000. In order to prove beyond question that gold was not escaping, an 8-in. Sturtevant exhaust fan was connected to the vertical stack and the gases filtered through muslin bags. The dust and fume collected from two separate tests of an hour's duration amounted in both cases to less than 5 lb., and assayed 1 oz. per ton.

The following anaylsis will give an idea of the baseness of the precipitate to be smelted:

	1911. Per cent.	1910. Per cent.	1909. Per cent.
Au	21.50	8.75	13.43
Ag	3.00	1.14	2.13
Cu	39.80	21.59	22.85
Pb	4.60	28.58	6.86
Zn	15.50	12.57	32.50
Cd	.....	0.94	.....
Fe	.....	0.18	.....
Mn	.....	Trace	.....
As	.....	0.06	.....
Sb	.....	0.11	.....
Bi	.....	0.02	.....
Ni	.....	0.08	.....
Co	.....	Trace	.....
Te	.....	2.12	.....
S	.....	1.38	.....
P	.....	0.017	.....
SiO_2	.....	1.54	.....
CO_2	.....	1.54	.....
Soluble alkaline salts	.....	0.74	.....
Moisture (105° C.)	.....	1.25	.....
Combined water	.....	3.19	.....

The furnaces are 'blown-in' in the following manner: A wood fire is built in the crucible, the blast turned on, and wood thrown in from the charging floor until the crucible and lead-well are cherry

red. A few charges of coke are added and the blast maintained until the whole charge is white-hot. The blast is then cut off and about 500 lb. of pig lead fed in to fill the crucible. The siphon is kept plugged with brasque in order to fill the crucible and float the ashes, charcoal, etc., which are raked out and subsequently returned to the furnace. When these have been removed the siphon is opened and blank charges of coke and slag are fed until the furnace is half full. This operation requires about 2 hours. By filling the furnace with these blank charges a bed is made for the first charge of briquettes, which prevents dusting. When the furnaces are ready, the following charge is fed:

	Lb.
Briquettes	160
Old slag	40
Borax	10
Cupel bottoms	10
Iron (oxidized)	5
Coke	25

With the exception of one run, when the slag contained 15% Zn, and become too pasty to work well, 24 hours is required to smelt 16,000 lb. of briquettes. The lead bullion contains approximately 20% gold and silver and 1% copper, which makes the lead-wells 'mushy.' In order to keep them open it is necessary to pass a hot iron rod through the siphon. The matte-fall is about one-third of the weight of the briquettes and is collected in a portable settler, from which the slag overflows into pots. The matte from the precipitate-run contains approximately 20% Pb, 50 oz. Au, and 200 oz. Ag per ton. This is stored until a sufficient quantity is on hand to make a separate run. The crucible of one furnace is then filled with brasque, and the furnace operated as a copper furnace. The separation of lead and matte is made in the portable settler. This operation reduces the lead in the matte to 10% and the gold to 3 oz. By roasting this product and leaching the copper and silver with H_2SO_4 it has been found possible to avoid shipping it, and a small plant will soon be erected for this purpose. When the slag becomes too 'zincy' for further use, it is either discarded or shipped to the smelter, according to the value.

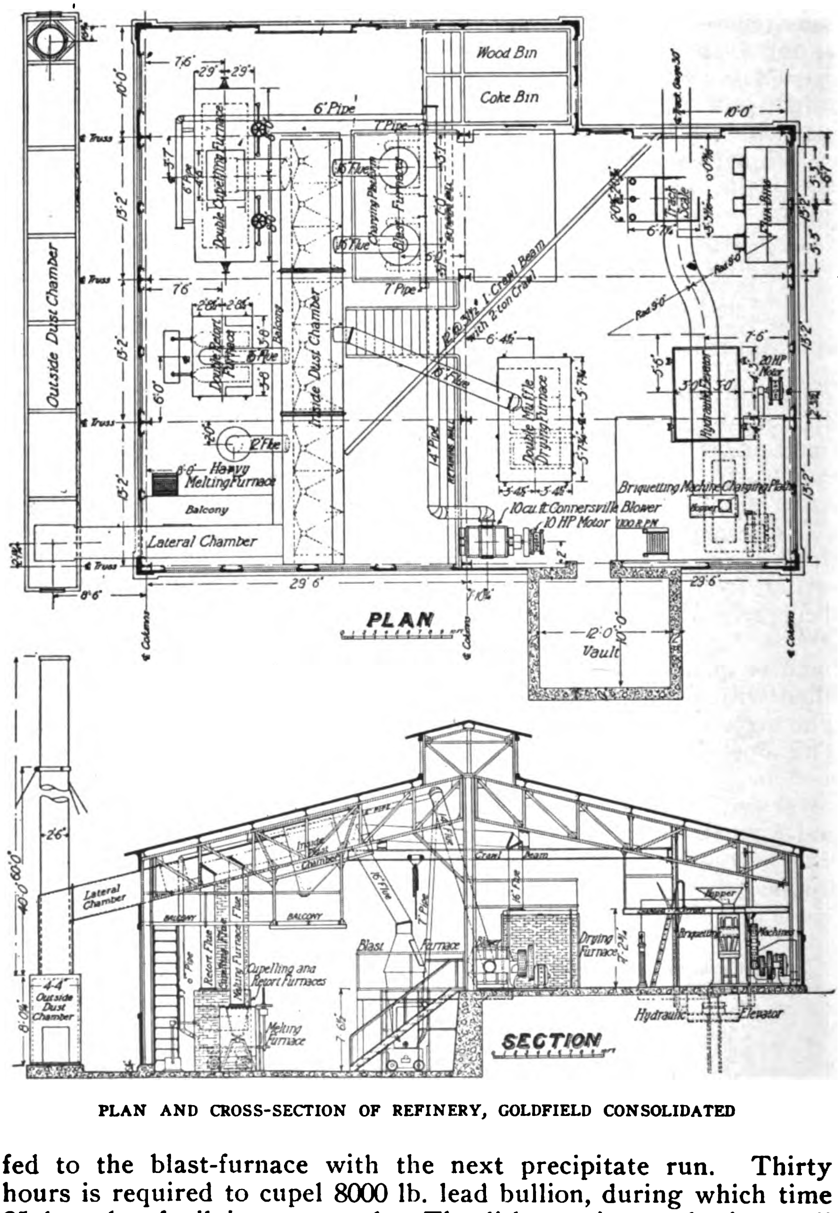
The composition of the slag is as follows:

	%
Insoluble	24.8
FeO	26.3
CaO	5.5
ZnO	12.3
Pb	6.5
Cu	1.3

Soluble silicates and borates undetermined; gold, 1.2 oz per ton.

Cupellation of the lead bullion is accomplished in a double English cupelling-furnace having removable tests, 3½ by 2½ ft. and 7 in. deep. The cupels are made of three parts cement and one part limestone; the latter is crushed to pass 6 mesh and the fine

screened through 20 mesh and discarded. They are seasoned for six weeks before using and last for one run, cupelling approximately 4000 lb. lead bullion, after which they are broken up and



PLAN AND CROSS-SECTION OF REFINERY, GOLDFIELD CONSOLIDATED

fed to the blast-furnace with the next precipitate run. Thirty hours is required to cupel 8000 lb. lead bullion, during which time 25 barrels of oil is consumed. The litharge is caught in small ladles and examined for beads of bullion before it is sent to the

grinding-room. Should any appear it is returned to the cupels. In finishing the cupellation it is necessary to add about 200 lb. of pig lead to each cupel in order to remove the remaining base. After the last matte and litharge have been floated off, the heat is raised, and the blast increased for 20 min. to oxidize the film of base which cannot be removed mechanically. When oxidation is complete, the gold bullion is granulated by pouring into metal tubs filled with water. The material is then dried and melted with nitre and borax in a No. 60 Steel-Harvey tilting furnace. The bars thus produced, averaging 930 gold and silver, are shipped to the Selby Smelting & Lead Co. for further refining. Approximately 25% of the total lead used is lost. The matte and slag account for the greater part of this and the cupellation losses for the rest. The litharge recovered from the cupels is broken in a 4 by 6-in. Dodge crusher and pulverized to 20 mesh in a set of 7 by 14-in. rolls and used for fluxing the precipitate from the next clean-up.

The cut on page 578 shows the refinery in plan and cross-section. In order to make it the more easily understood, the names of the more important features have been placed on the drawing, and it will therefore not be necessary to give any further description. The building is a steel frame covered with wire netting and portland cement plaster. Concrete was used liberally in constructing floors and foundations, and it is as nearly fireproof as such a building may be made.

It is not intended to give the impression that the process is perfect, and that the usual amount of 'grief' has not been passed through in getting started; but when the character of the product is taken into consideration, the results are very satisfactory, as can be seen from the accompanying tables:

Cost of Melting

	Blast-Furnaces and Cupellation.	Acid Treatment and Tilting-Furnaces.	
	1911.	1910.	1909.
Labor	4.2	5.0	6.1
Supplies	5.2	13.2	14.0
Power	0.1	0.1	0.2
	9.5	18.3	20.3

Concentrate Treatment

Up to June, 1908, very little progress had been made in evolving a process for treating the high-grade concentrate which would be produced at the mill. The product from the Combination mill of this company had been shipped to the smelters until J. H. Mackenzie assumed the management. The freight and treatment charges were so excessively high at that time that he decided to store the concentrate, pending the solution of the problem of treating it. The reports to the management by the former metal-

Consumption of Chemicals and Cost of Converting $KAuCN_2$ Into Fine Gold

	Per Ton Ore Milled.			Per Base Oz. Bullion.			Per Fine Oz. Gold.		
	1909.	1910.	1911.	1909.	1910.	1911.	1909.	1910.	1911.
			Fine	354	378	850	Au		
				61	39	80	Ag		
Consumption.	Lb.	Lb.	Lb.				Lb.	Lb.	Lb.
KCN	1.60	2.61	3.12	0.508	0.77	1.90	1.4	2.0	2.31
Lime	8.72	8.49	8.55	2.70	2.50	5.20	7.61	6.5	6.5
Zinc dust	1.18	1.02	0.50	0.37	0.30	0.30	1.06	0.76	0.38½
Lead acetate	0.65	0.74½	0.58	0.26	0.225	0.37	0.73	0.57	0.44
Litharge			0.165			0.10			0.118
Pig lead			0.07			0.043			0.05
Lb. dry ppt. produced	1.00	0.92	0.52	0.313	0.254	0.32	0.876	0.70	0.38
Cost of	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.	Cts.
Precipitating	12.0	11.0	7.3	3.8	3.25	4.5	10.7	8.4	5.4
Melting	20.3	18.3	9.5	6.4	5.4	5.8	18.1	14.0	7.0
Shipping bullion	8.4	8.8	8.1	2.6	2.6	5.0	7.5	6.7	6.0
Mint charge for refining base bullion	11.8	11.03	6.7	3.7	3.25	4.1	10.6	8.4	4.9
Total cost ppt. melt. and marketing	52.5	49.13	31.6	16.5	14.50	19.4	46.9	37.5	23.3

lurgists of this company stated that roasting would be necessary for economical treatment, although the results of tests made did not prove conclusively that the concentrate could be treated successfully in this way.

As can be seen from the flow-sheet given concentration follows tube-milling. Roasting the -200-mesh concentrate, containing 30% sulphur and 20 oz. gold, to put it mildly, seems dangerous. In addition, roasting is not necessary. Much of the visible gold in the ore from this company's mines, although apparently free, is so coated that it cannot be recovered by amalgamation, and is only slowly soluble in cyanide solution. During the summer of 1908, when engaged in experimental work on the concentrate, an acid wash was applied for the purpose of removing this coating, in order to amalgamate the coarse gold if possible. It was found that this acid wash did remove the coating, and that much more gold could be recovered by amalgamating the concentrate after an acid wash had been applied. It seemed probable from this result that cyanidation of the raw concentrate would be facilitated by the same process, since it was not reasonable to suppose that only the coarser particles of gold were thus coated. Experiments were made on a laboratory scale and the results were astonishing. Approximately 90% of the gold in 20-oz. concentrate was dissolved in 8 hr. contact with a 4-lb. solution of cyanide, after a preliminary acid wash. The results were corroborated by larger tests, and were so encouraging, the management leased the old Kinkead mill for the purpose of treating the accumulation of concentrate from the Combination mill. During the summer of 1908 approximately 300 tons was treated and yielded 94% of the gold content, at a cost which made it seem advisable to build a plant of similar type at the 100-stamp mill. It was found during experimental work, that the solutions became inert after 8 hr. contact, and that it would be necessary to remove them and re-treat the concentrate with a solution freshly precipitated and freed by reducing agents. As originally designed and operated, the process of cyanide treatment after the acid wash consisted in agitating for 8-hr. periods in Pachuca vats, passing the pulp at the end of these periods through Dorr continuous thickeners; the clear overflow passing to the precipitating department and the thickened pulp to a second Pachuca vat, where a regenerated solution was added. This arrangement proved expensive, since it necessitated pumping the concentrate each time, and was abandoned. The process in use now consists in agitating the concentrate for 8-hr. periods, and settling and decanting in the Pachucas, and is very satisfactory. During the first year's run, much difficulty was experienced in evolving a mechanical stirrer for the pulp during the acid wash and the subsequent water washes.

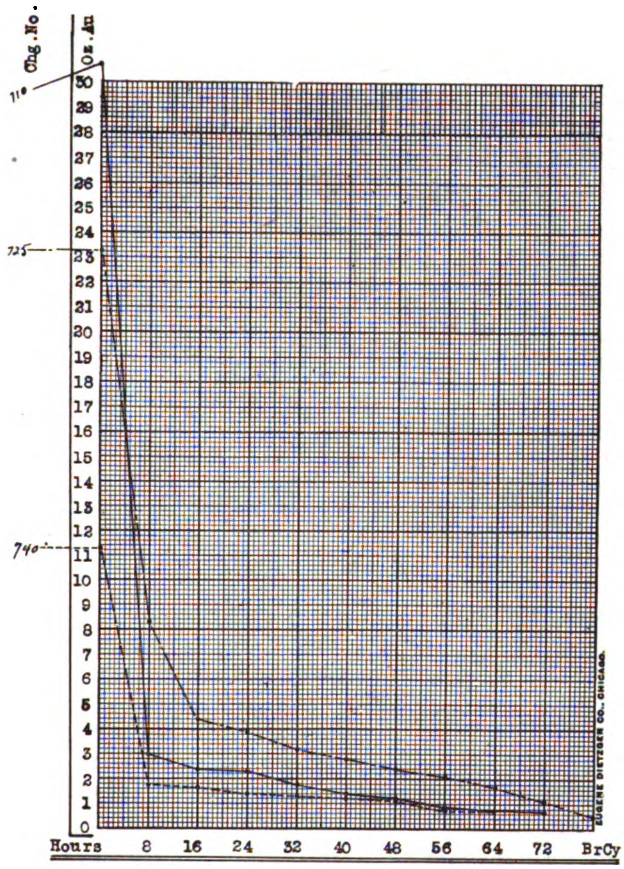
The concentrate from the mill gravitates through wooden launders to four 48 in. by 16 ft. amalgamating tables. The last section of these tables is covered with carpet for removing the coarser

particles of gold, which do not amalgamate. The recovery by the combined process of amalgamating and carpeting amounts to approximately 35% of the value of the concentrate, or 25% of the value of the ore. Of this amount, 10% is contained in the carpet concentrate, which is briquetted with the precipitate from the cyanide plant, and supplies the necessary sulphur for matting the copper in it. The pulp from the plates runs to three 10 by 20-ft. redwood collecting and agitating vats, fitted with the above-described adjustable stirrer. When a charge is being collected, the vertical shaft is pulled up by means of the chain-block, so that there is no difficulty in starting it up. Each vat holds one day's run, amounting at present to 50 tons of dry concentrate. After the charge has been collected, the water is decanted, leaving a pulp containing approximately 50% moisture. The agitator is started up and gradually lowered until the whole charge is in motion. To this is added 66°B. sulphuric acid in the proportion of 20 lb. per ton of concentrate, and agitation continued for 8 hours. The vat is then filled with water, after which the agitator is shut off and raised, and the charge allowed to settle. The clear solution is decanted and runs to a special tank in the mill, is neutralized there, and used in the mill for crushing purposes. Two more water washes are added in like manner, equivalent to 6 parts by weight of water per ton of dry concentrate. The last water wash is decanted as closely as possible to avoid subsequent dilution of the cyanide solution, lime added in quantities to raise the alkalinity to ½ lb. in terms of CaO, the agitator started, and when the charge is in motion, 1 lb. of lead acetate per ton of concentrate is dissolved in water and added to the pulp, which is kept in motion until time to pump to the Pachucas.

It has been found most necessary here not to let the concentrate be in contact for any length of time with a weak solution of cyanide. For this reason the amount of cyanide required to bring the solution on the charge to 4.5 lb. KCN per ton, is first dissolved in the Pachuca agitator and the charge pumped into this. Exactly what effect a dilute solution of cyanide has on a heavy sulphide ore is not known here, but that the effect is deleterious has been demonstrated beyond question by the following results. In order to avoid the small mechanical loss of cyanide due to neutralizing the concentrate with milk of lime, it was decided to slack the lime with solution from the mill. This was done, and titration of the solution in the neutralizing tank showed a content of from 0.2 to 0.3 lb. KCN per ton. This practice was continued for nearly two weeks, with the result that the assay value in the tailing was doubled. Immediately on discontinuing the KCN solution in the neutralizing tank, the extraction came up to normal. This has been tried on different charges since, with the same result each time.

The same effects have been noted when treating a heavy silver sulphide ore which had been crushed in water, and then transferred to the strong solution in the agitators. When this same ore was crushed in the mill solution, and allowed to stand in the dewatering

tanks in dilute cyanide solution, before being transferred to the agitators, the extraction was materially reduced and the strength of the solution used for agitating had to be increased in order to approximate the same results obtained by cyanidation after crushing in water. It is not to be understood that crushing silver sulphide in water is believed to be the most economical plant for low-



EXTRACTION CHART, GOLDFIELD CONSOLIDATED

grade ores, but the instance is cited to show that contact with a weak solution of cyanide is deleterious to the treatment of concentrate and ores containing large quantities of sulphides. At the end of the fourth hour, in alternate periods, peroxide of sodium is added to the charge, which increases the activity of the solution and reduces the time of treatment. One-tenth pound lead acetate per ton of dry concentrate is added to each solution at the beginning of the period.

The accompanying chart, showing the extraction graphically by periods, is representative of the various conditions and grades of concentrate. For two years the extraction of gold from the pulp in the Pachuca agitators has averaged 93%, and the total extraction by amalgamating, carpeting, and cyaniding has averaged 95.23 per cent.

A Kelly filter-press (type B) containing 400 sq. ft. of filter surface is used for filtering the concentrate. It was found necessary to set this machine at a much greater inclination than is required for ordinary slime, since the heavy pulp has a tendency to pack in the bottom of the cylinder which prevents the carriage from running out freely. The labor of one operator and two trammers is required to filter and dispose of 50 tons of concentrate in 8 hours. On this material the capacity of the Kelly press is equivalent to 750 lb. concentrate and 1200 lb. solution per square foot of filter per day of 24 hours.

The cost of cyaniding the concentrate is as follows:

Labor	$0.93
Supplies	4.44
Power	0.48
Total	$5.85

Summary of Recovery and Cost

Recovery:	1911. %	1910. %	1909. %
By amalgamation	17.55	15.38	10.60
By concentration	53.93	56.86	49.20
By cyanidation	22.56	22.03	32.80
Total	94.04	94.27	92.60

Cost per ton milled:	1911. %	1910. %	1909. %
Crushing-conveying	0.040	0.071	0.053
Sampling	.....	0.003	0.021
Stamping	0.134	0.174	0.195
Elevating-separating	0.022	0.023	0.021
Chilean milling	0.097	0.095	.....
Tube-milling	0.177	0.187	0.206
Concentrating	0.057	0.059	0.062
Amalgamating	0.025	0.033	0.058
Neutralizing	0.045	0.046	0.046
Settling	0.053	0.055	0.055
Agitating	0.604	0.561	0.503
Experimental	0.102	.....	.....
Filtering-discharging	0.068	0.084	0.093
Assaying	0.046	0.045	0.063
Precipitating	0.074	0.110	0.120
Refining	0.098	0.183	0.203

Cost per ton milled:	1911. %	1910. %	1909. %
Water service	0.098	0.112	0.110
Surface and plant	0.007	0.011	0.015
Steam heating	0.056	0.032	0.023
Watchmen	0.042	0.049	0.031
Storehouse and office	0.022	0.028	0.031
Stable	0.004	0.004	0.005
Lighting	0.021	0.018	0.019
Superintendence	0.062	0.067	0.082
General expense	0.012	0.012	0.009
Mill tools	0.002	0.003	0.005
Mechanical department	0.001	0.004	0.008
Electrical department	0.034	0.026	0.007
Return water service	0.010	.....	.....
Fire loss (machine-shop)	.....	0.026	.....
Mill total	2.013	2.131	2.040
Concentrate plant total	0.381	0.312	0.276
Total mill and conc. plant	2.394	2.433	2.316
*Mill operation	1.859	1.828	1.820
*Mill repairs	0.154	0.283	0.220
*Concentration plant operation	0.371	0.298	0.256
*Concentration plant repairs	0.010	0.014	0.020

*Included in the above, but given for additional information.

On page 586 is given a chart of the power-load analysis by tonnage which is sufficiently self-explanatory. This is the chart mentioned upon page 552, where the cost of the average power load, 1.73 hp. per ton milled, is given as 32c. per ton.

(Editorial, May 27, 1911)

Volatilization of gold at high temperatures is a well known fact, but there is a tendency to exaggerate its importance. In the roasting of gold precipitates and telluride ores even the mechanical losses may be kept very low by careful conduct of the operation. This is emphasized by the results cited by Mr. J. W. Hutchinson in the part of his description of the operation of the Goldfield Consolidated mill which we print this week. In smelting the briquetted precipitate from the refinery in a small blast-furnace, it was found that the loss in the flue-dust amounted to one-eighth of one per cent of the precious metals, practically all of which was afterward recovered. The fine material caught by filtering the escaping gases through bags had a value of only $20 per ton, while the ordinary flue-dust had a value of $2500 per ton. The recovery is therefore complete from a commercial standpoint. In roasting Cripple Creek ores the usual loss in the flue-dust is one per cent, most of which is afterward recovered.

Rated H.P.	R.P.M.	Make	Series No.		Peak K.H.P.	Ave. H.P. per 24 hr. day	Ave. K.W. per 24 hr. day	Ave. K.W. hr. per 24 hr. day	Ave. K.W. hr. per 30 da. Month	K.W. hr. per ton Basis 20,000 tons per Mo.	
				CRUSHING, SAMPLING, STAMPING AND REGRINDING.							
150	570	Bullock	42733	Crusher	75.5	23.01	17.17	411.97	12530.81	0.6265	
75	680	do	40468	Elevator and Samplers	45.6	16.40	12.23	293.62	8931.11	0.4466	
15	1130	do	32834	Distributor	9.3	5.40	2.54	60.87	1851.57	0.0926	18.4989 K.W. hr. Per ton for CRUSHING, SAMPLING, STAMPING AND REGRINDING.
50	680	do	42805	Stamps	242.5	231.44	172.65	4143.70	126037.57	6.3019	
50	680	do	42761								
50	680	do	42756								
50	680	do	42735								
50	680	do	42755								
200	570	do	50932	Tube Mills	387.6	356.73	266.12	6386.89	194268.04	9.7134	
200	570	do	50887								
40	850	do	42783	Mill-Water Pumps	66.0	48.40	36.11	866.55	26357.67	1.3179	
40	850	do	42823								
				CONCENTRATING.							
30	850	do	40556	Deister Slimers	44.0	41.18	30.72	737.29	22425.82	1.1213	1.1213 K.W. hr. Per ton for CONCENTRATING.
30	850	do	40555								
				CYANIDING.							
20	1130	do	44321	Stock Pulp Mechanical Agitators	25.6	21.00	15.67	375.98	11436.18	0.5718	
10	850	do	44559	Excess Pulp do. do.	12.8	6.18	4.61	110.65	3365.52	0.1683	
30	850	do	40554	Pulp Pumps Excess to Stock Tanks	35.0	10.02	7.47	179.40	5456.68	0.2728	
30	850	do	40701	Circulating Pumps Pachuca Agitators	49.0	26.80	19.99	479.83	14594.74	0.7297	
30	850	do	44425	Circulating Pump Butters Filters	15.6	6.90	5.15	123.54	3757.60	0.1879	
15	850	do	44131	Vacuum Pumps do. do.	15.0	9.34	6.97	167.22	5086.35	0.2543	5.5631 K.W. hr. Per ton for CYANIDING
15	850	do	44203								
20	850	do	40576	Clarifying Presses Pump	24.5	22.01	16.42	394.07	11986.24	0.5993	
20	1130	do	44336	Circulating Pumps Wash Water	28.5	19.43	14.49	347.88	10581.20	0.5291	
30	1130	do	44442								
15	850	do	44136	Pumps Gold Solution to Merrill Presses	25.6	22.07	16.46	395.14	12018.87	0.6009	
15	850	do	44139								
7.5	1200	General Electric	104021	Lime Crusher	2.2	2.00	1.49	35.81	1089.16	0.0544	
				Oil Heater	14.8	5.00	3.73	89.52	2722.90	0.1361	
75	684	Westinghouse	504416	Air Compressors	140.0	107.00	79.82	1915.73	58270.06	2.9135	
75	720	General Electric	108658								
15	850	Westinghouse	410418	Tramway	12.0	0.13	0.10	2.33	70.81	0.0035	2.3097 K.W. hr. Per ton for CONCENTRATE TREATMENT.
				CONCENTRATE TREATMENT.							
50	850	Bullock	30419	Agitators	30.0	18.20	13.58	325.85	9911.36	0.4956	
20	1200	General Electric	174100	Kelly Press	19.0	5.23	3.90	93.64	2848.17	0.1424	
20	1130	Bullock	44353	Pachucas	34.5	7.83	5.84	140.19	4264.07	0.2132	
20	850	do	40605								
				REFINING.							
10	1130	Bullock	42074	Agitators, Johnson Presses, Slag Crusher	5.5	2.80	2.09	50.13	1524.82	0.0762	0.1021 K.W. hr. Per ton for REFINING
2	1800	General Electric	135530	Acid Wash Pump	1.7	0.30	0.22	5.37	163.37	0.0082	
3	1800	do	126509	Fumes Collector	1.3	0.65	0.49	11.64	353.98	0.0177	
				GENERAL.							
10	1130	Bullock	40593	Machine Shop	12.0	1.20	0.90	21.48	653.50	0.0327	1.2940 K.W. hr. Per ton for SHOPS & LIGHTING.
3	1800	General Electric	P.116180	Carpenter Shop	4.0	0.30	0.22	5.37	163.37	0.0082	
				Lighting	78.0	46.02	34.33	823.94	25061.56	1.2531	
1630.5	Total Installed Horse Power in Motors				1454.9	1060.97	791.48	18995.61	577783.10	28.8891	

POWER ANALYSIS BY TONNAGE, GOLDFIELD CONSOLIDATED MILL

OPERATING COSTS AT THE GOLDFIELD CONSOLIDATED MILL

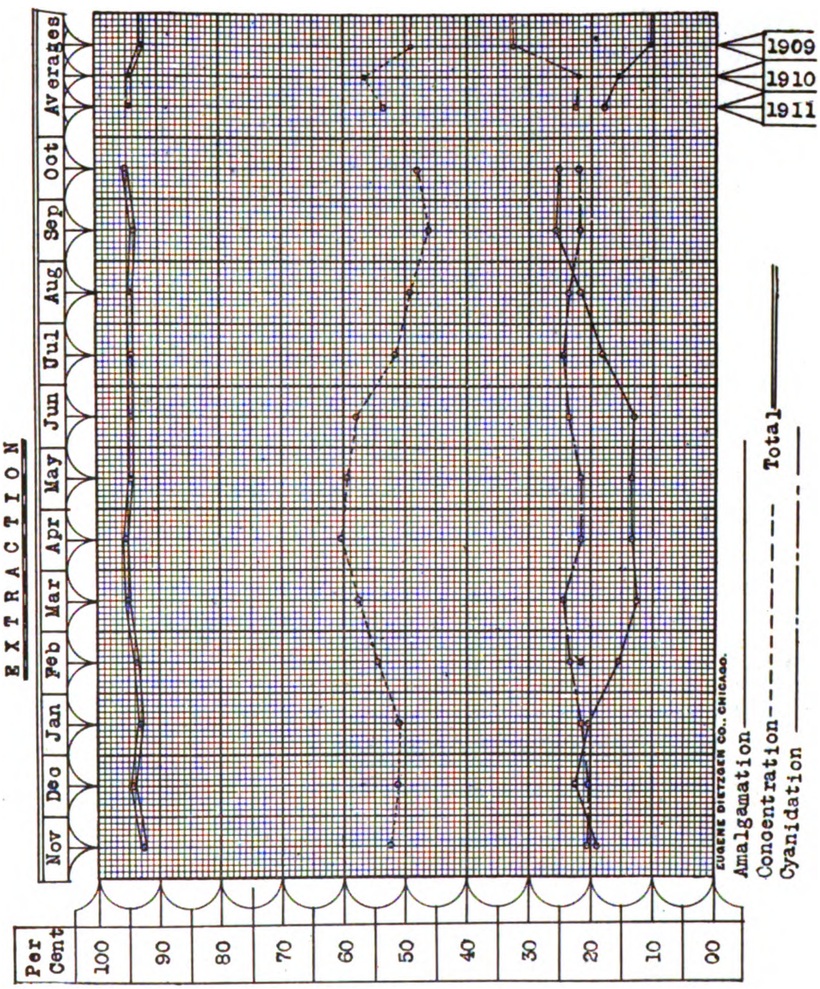
(January 20, 1912)

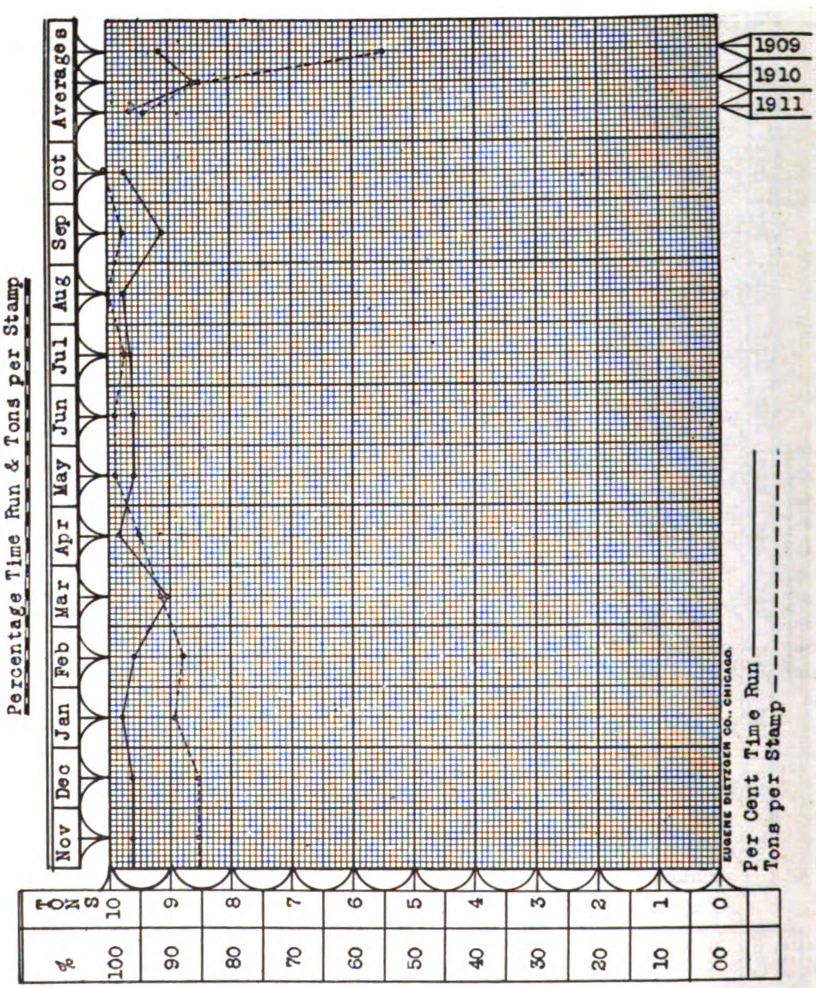
A detailed description of the mill used for treating the ores of the Goldfield Consolidated Mines Co., at Goldfield, Nevada, was published in the *Mining and Scientific Press,* early in 1911*. In brief, the process involves three-stage reduction by means of stamps, Chilean mills, and tube-mills, followed by concentration and cyanidation, using Pachuca agitators, Butters vacuum-filters, and Merrill zinc dust precipitation. The concentrate is treated by amalgamation and a special cyanide, or bromo-cyanide, process. Detailed figures covering operating costs for the year ended October 30, 1911, are given below, being taken from an admirable general statement of costs at the property, compiled by the superintendent, J. W. Hutchinson, and generously furnished by him to his brother cyaniders. Owing to limitations of space, only a few of the 32 charts furnished by Mr. Hutchinson have been reproduced. The figures are perhaps self-explanatory, though in considering them it is well to recall that Goldfield is a district of high wages, $3.50 to $5 for 8 hours' work, and of high power costs, since electricity must be brought from the mountains. The cost in fact is $6 per horse-power-month, based upon 90% of the peak load. When these figures are compared with the $15 per year rate obtaining at some plants in Norway, or even the rates in certain American mining districts, it will be recognized that the total must inevitably be high at Goldfield. Water amounts to 220 gal. per ton of ore milled and costs 50c. per thousand gallons. It should also be noted that the ore milled is high in grade, averaging approximately $40 per ton through two years, and the amount treated is large, about 28,000 tons per month. The actual amount treated in any month may be calculated from one of figures below, remembering that the mill includes 100 stamps. All the figures are based upon total tons treated. For example, the Chilean mill costs, as will be noted, ranged from 8 to 12½c. per ton, but to get a figure for comparison with costs elsewhere it is necessary to remember that only a part of the total tonnage goes through the Chilean mills. For the actual amount, and other data, readers are referred to Mr. Hutchinson's own account of the mill, already cited. Figures for 1909 to 1911 inclusive are reproduced below.

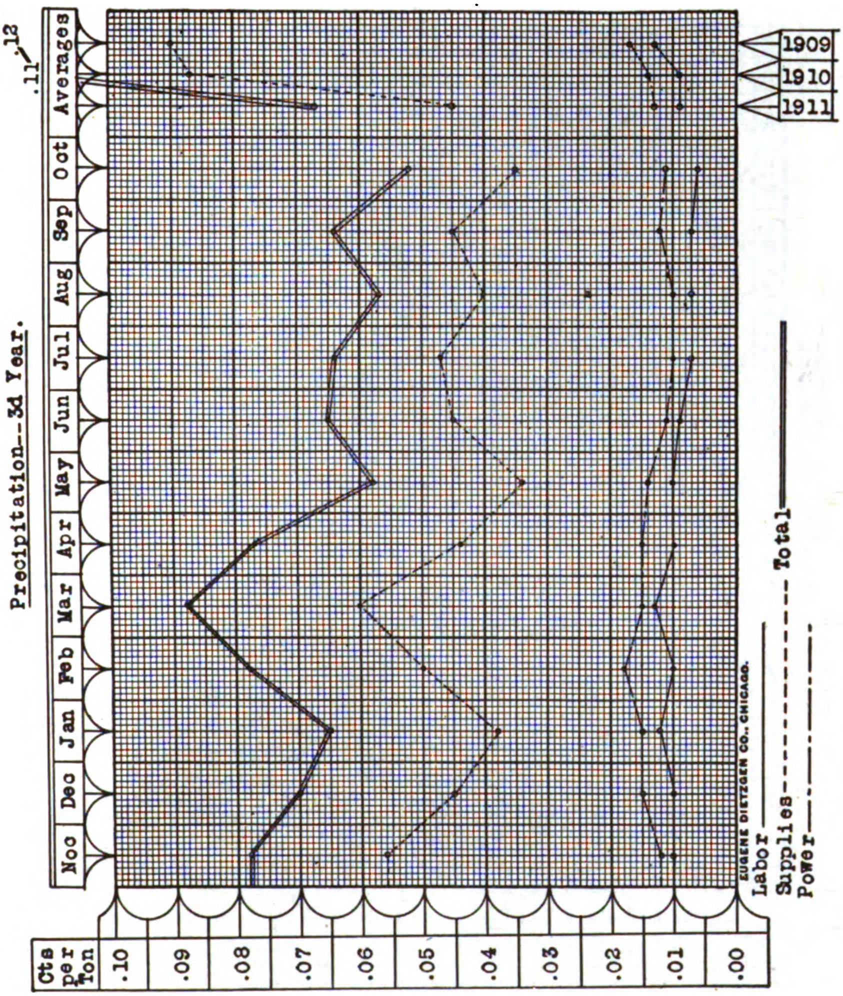
	1911.	1910.	1909.
Cost per ton milled:	%	%	%
Crushing-conveying	0.040	0.071	0.053
Sampling	.....	0.003	0.021
Stamping	0.134	0.174	0.195
Elevating-separating	0.022	0.023	0.021
Chilean milling	0.097	0.095	.....
Tube-milling	0.177	0.187	0.206

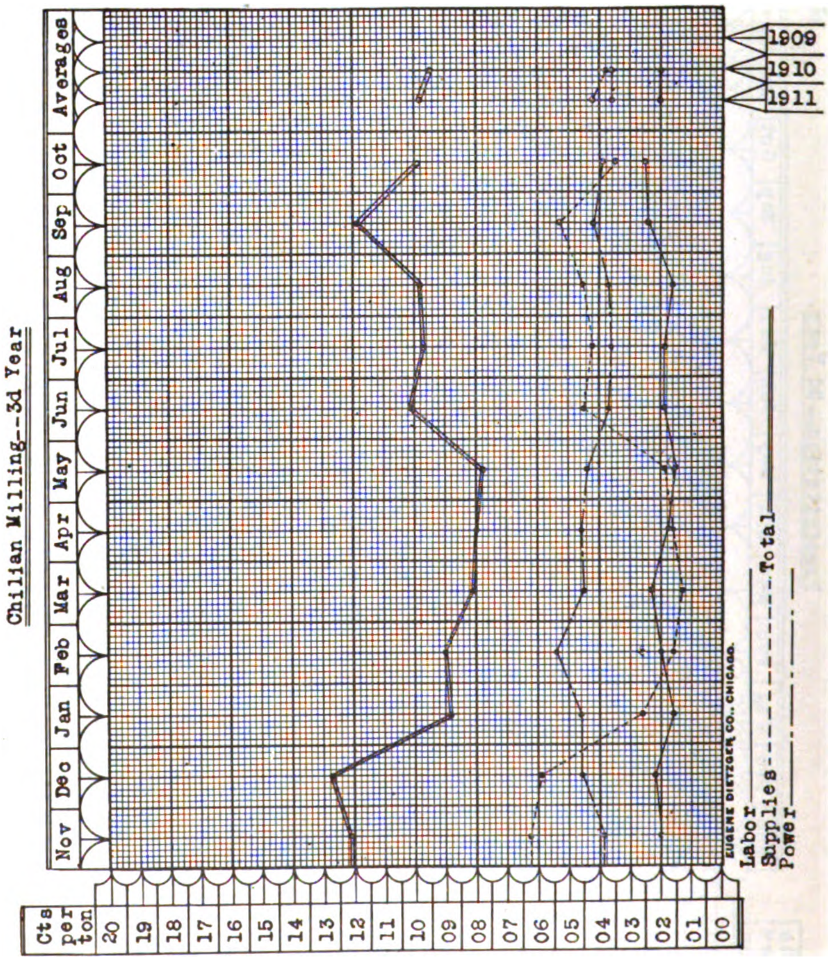
*'Goldfield Consolidated Mill Operation,' by J. W. Hutchinson, *Mining and Scientific Press,* May 6, 13, 20, 27, June 10, 1911. (Reprinted Page 547).

Cost per ton milled:	1911. %	1910. %	1909. %
Concentrating	0.057	0.059	0.062
Amalgamating	0.025	0.033	0.058
Neutralizing	0.045	0.046	0.046
Settling	0.053	0.055	0.055
Agitating	0.604	0.561	0.503
Experimental	0.102	.....	.....
Filtering-discharging	0.068	0.084	0.093
Assaying	0.046	0.045	0.063
Precipitating	0.074	0.110	0.120
Refining	0.098	0.183	0.203
Water service	0.098	0.112	0.110
Surface and plant	0.007	0.011	0.015
Steam heating	0.056	0.032	0.023
Watchmen	0.042	0.049	0.031
Storehouse and office	0.022	0.028	0.027
Stable	0.004	0.004	0.005
Lighting	0.021	0.018	0.019
Superintendence	0.062	0.067	0.082
General expense	0.012	0.012	0.009
Mill tools	0.002	0.003	0.005
Mechanical department	0.001	0.004	0.008
Electrical department	0.034	0.026	0.007
Return water service	0.010	.....	.....
Fire loss (machine-shop)	.....	0.026	.....
Mill total	2.013	2.131	2.040
Concentrate plant total	0.381	0.312	0.276
Total mill and conc. plant	2.394	2.433	2.316

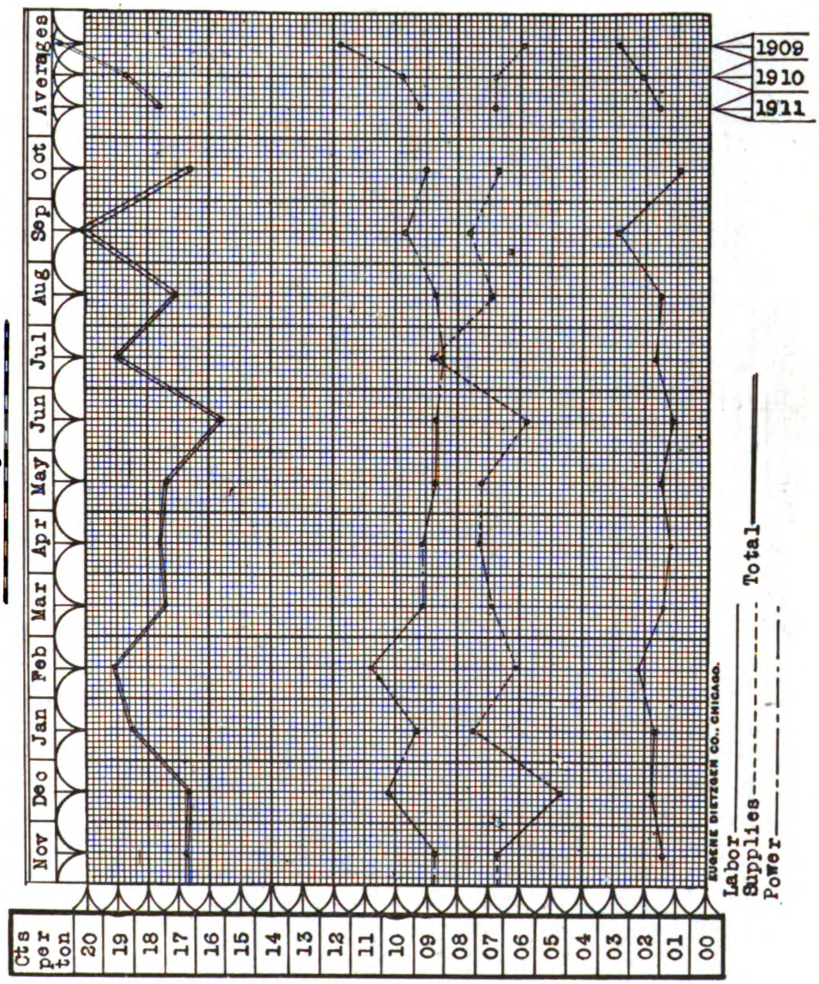

EXTRACTION
Nov
Dec
Jan
Feb
Mar
Apr
May
Jun
Jul
Aug
Sep
Oct
Averages
1909
1910
1911
Per Cent
100
90
80
70
60
50
40
30
20
10
00
EUGENE DIETZGEN CO., CHICAGO.
Amalgamation
Concentration
Cyanidation
Total

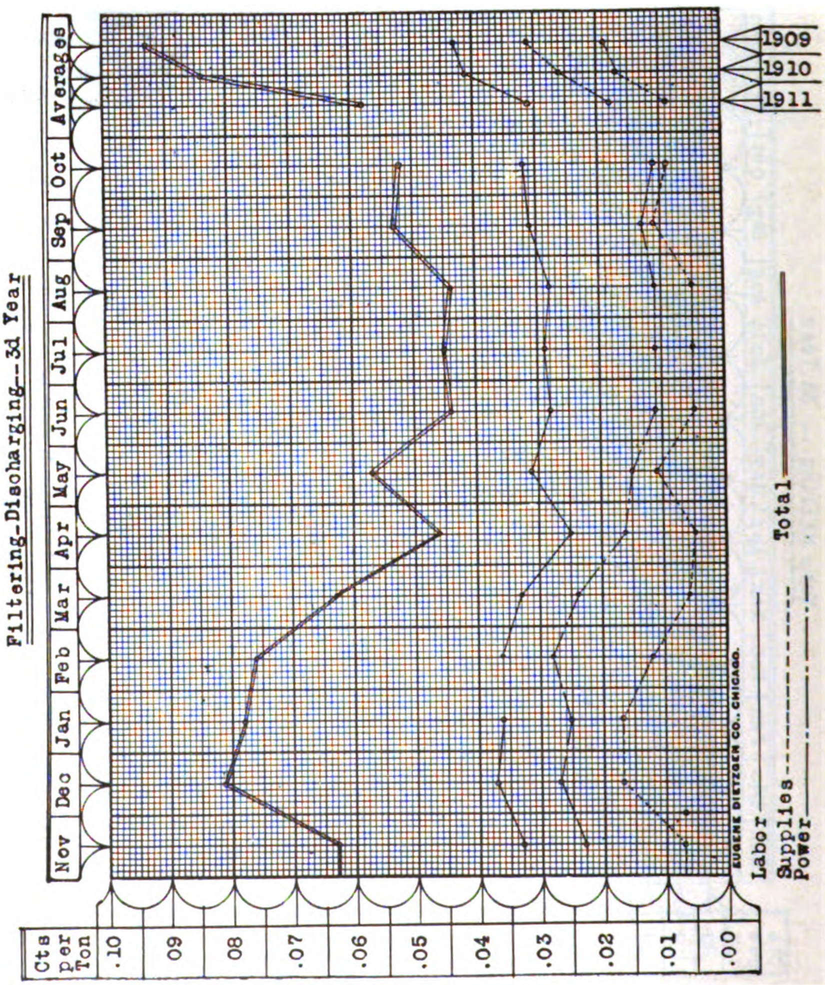

Percentage Time Run & Tons per Stamp
Nov
Dec
Jan
Feb
Mar
Apr
May
Jun
Jul
Aug
Sep
Oct
Averages
1909
1910
1911
TONS
%
10
9
8
7
6
5
4
3
2
1
0
100
90
80
70
60
50
40
30
20
10
00
EUGENE DIETZGEN CO., CHICAGO.
Per Cent Time Run
Tons per Stamp

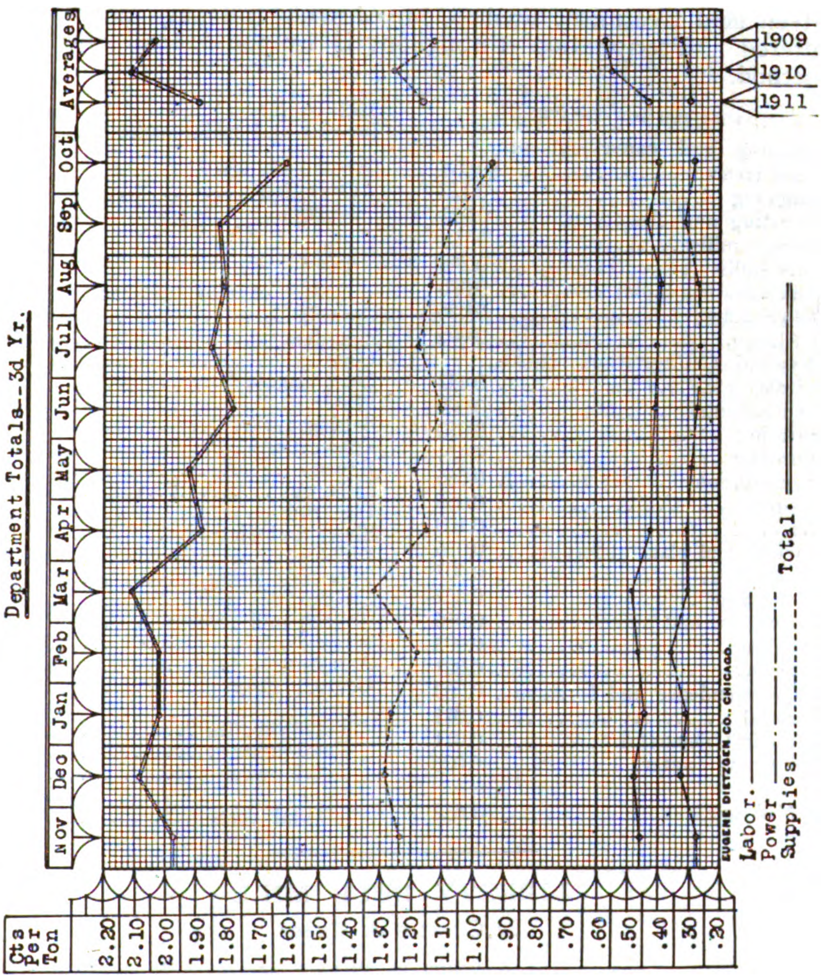

Precipitation--3d Year.
Nov Dec Jan Feb Mar Apr May Jun Jul Aug Sep Oct Averages
Cts per Ton
.10 .09 .08 .07 .06 .05 .04 .03 .02 .01 .00
1909
1910
1911
.11
.12
EUGENE DIETZGEN CO., CHICAGO.
Labor
Supplies
Power
Total


Chilian Milling--3d Year
Nov
Dec
Jan
Feb
Mar
Apr
May
Jun
Jul
Aug
Sep
Oct
Averages
Cts per ton
20
19
18
17
16
15
14
13
12
11
10
09
08
07
06
05
04
03
02
01
00
1909
1910
1911
EUGENE DIETZGEN CO., CHICAGO.
Labor
Supplies
Power
Total

Tube Milling -- 3d Year




Filtering-Discharging--3d Year
Nov
Dec
Jan
Feb
Mar
Apr
May
Jun
Jul
Aug
Sep
Oct
Averages
1909
1910
1911
Cts per Ton
.10
.09
.08
.07
.06
.05
.04
.03
.02
.01
.00
EUGENE DIETZGEN CO., CHICAGO.
Labor
Supplies
Power
Total


Department Totals--3d Yr.
Nov
Dec
Jan
Feb
Mar
Apr
May
Jun
Jul
Aug
Sep
Oct
Averages
1909
1910
1911
Cts Per Ton
2.20
2.10
2.00
1.90
1.80
1.70
1.60
1.50
1.40
1.30
1.20
1.10
1.00
.90
.80
.70
.60
.50
.40
.30
.20
EUGENE DIETZGEN CO., CHICAGO.
Labor.
Power
Supplies
Total.

(December 28. 1912)

The following costs have been supplied by J. W. Hutchinson, metallurgist of the Goldfield Consolidated mill, which consists of 100 stamps crushing 1000 tons per day, followed by classifiers, Chilean mills, tube-mills, concentrators, agitators, and vacuum-filters; while the concentrate is agitated raw with cyanide, roasted, given acid treatment, washed, mixed with mill pulp, and finally filtered. Precipitation throughout is by the Merrill zinc dust method.

COST PER TON TREATED

Crushing and conveying, per ton treated	$0.034
Stamping	0.133
Elevating and classifying	0.023
Chilean mills	0.104
Tube-mills	0.208
Concentrating tables	0.070
Neutralizing	0.054
Settling	0.065
Dissolution (agitation, chemicals, etc.)	0.453
Experimental	0.009
Filtering	0.050
Assaying	0.024
Precipitation	0.056
Refining	0.055
Water	0.108
Surface and plant generally	0.016
Steam heat	0.007
Watchmen	0.032
Storehouse and office	0.016
Stables	0.003
Lighting	0.021
Superintendence and foremen	0.063
General	0.011
Electrical department	0.011
Returning water	0.026
Mill total	$1.652

Mill operation	$1.488
Repairs and maintenance	0.164
Concentrate plant:	
Operation	0.377
Repairs	0.033
Total	$2.062

COST OF SUPPLIES

Cyanide	$0.328
Zinc-dust	0.034
Lime	0.091
Lead acetate	0.058
Water	0.108
Belting	0.013
Lubrication	0.007
Borax	0.006
Litharge	0.009
Pig lead	0.004
Shoes and dies	0.023
Pebbles	0.060
Tube-mill lining	0.019
Chilean mill steel	0.022
Chilean mill screens	0.006
Filter cloth	0.003
Assaying	0.021
General stores	0.121
Total supplies	$0.933
Total labor	0.388
Total power	0.332
Total costs	$1.652

LIBERTY BELL MILL

By Charles A. Chase

(June 24, 1911)

*The metallurgical practice here has frequently been described, and I will try not to dwell on well known features. The accompanying flow-sheet and ground-plan need no explanation, showing as they do the proper relations between the following units of equipment:

*A portion of a paper to be presented at the San Francisco meeting of the American Institute of Mining Engineers.

1. Eighty 850-lb. stamps with suspended Challenge feeders.
2. Sixteen copper amalgamating-tables, each 8 by 4 ft., with three 1-in. drops.
3. Four Richards vortex (hindered settling), 3-spigot classifiers.
4. Eighteen Wilfiey tables and 10 Deister No. 3 tables.
5. Three Abbé pattern, 5 by 22-ft. tube-mills, the feed thickened by Dorr classifiers, or diaphragm cones.
6. Eight amalgamating-tables, of the size given above.
7. Nine Dorr continuous settlers, 33 by 11 feet.
8. Six Hendryx type agitators, 17 by 11 ft., above the 45° cone.
9. One equalizer vat, 20 by 15 feet.
10. Moore filter-plant, 7 vats, each 9 by 27 ft. in area, and 8½ ft. to the coning.
11. Zinc-shaving precipitation-plant, capacity 1200 cu. ft. of zinc.

The stamp-battery was originally built on wood blocks, with the usual framing and front horizontal drive from clutch-pulleys on the line-shaft. The wood foundations have been replaced by concrete, and the framing is simpler. The post rests on the concrete, only a piece of 6-ply Gandy, or similar, belting intervening. The results have been perfect. Ten stamps have heavy Allis-Chalmers anvil-blocks, the other lighter Denver Engineering Works 'sub-bases.' There is no apparent difference in results, and the lighter construction is cheaper. Globe stem-guides have been reasonably satisfactory through many years, but are now being replaced by the simpler and stronger Pacific guides. Shoes, boss-heads, and tappets are of chrome steel. The cams are of the Allis-Chalmers Blanton pattern. This Blanton fastener is also used for the bull-wheel. Dies are of cast iron, from the local foundry, containing a large percentage of steel scrap.

The horizontal battery-drive through clutch-pulleys was unsatisfactory and solid pulleys were substituted, it being cheaper and easier to cut an occasional belt, in case of desiring to stop a 10-stamp section for considerable repairs, than to maintain the clutches. The feeders are operated by the feeder-wheel mechanism, a good device, patented by the mill foreman in 1900. Battery screens have recently been of two patterns; 14 by 14 mesh, No. 22 wire, aperture 0.043 in.; and 16 by 3 mesh Ton-cap, 0.039 aperture, which yields a finer product and a larger tonnage than the former, having heavier wire and the same aperture blinded. It seems that the wire must be light enough to spring readily under the impact of the splash. A screen analysis of battery pulp shows: On 40 mesh, 24.4%; on 60 mesh, 10.9%; on 80 mesh, 5.9%; on 100 mesh, 6.3%; on 200 mesh, 9.6%; through 200 mesh, 42.9%. Some 38% of the battery pulp is flocculent. The power charged to the battery amounts to 160 horse-power.

The ore is stamped in cyanide solution. The recovery by amalgamation is materially less than in previous years of water-

amalgamation; it is more expensive in both labor and material, and requires more skill, and the consumption of copper is considerable. Muntz-metal, which has proved a satisfactory substitute elsewhere, has not been successfully adopted here, taking on a hard, glassy surface. The plates are kept rather wet, and any drip of quick is caught in a trap. From a month's run the results of amalgamation were: from battery-plates, 80% of all amalgam, yielding 29% bullion, 0.408 Au, 0.551 Ag; from tube-mill plates, 20% of all amalgam, yielding 23% bullion, 0.153 Au, 0.825 Ag; the first plates have a grade of 2¼ in. per foot, and the second plates 1⅝ inches.

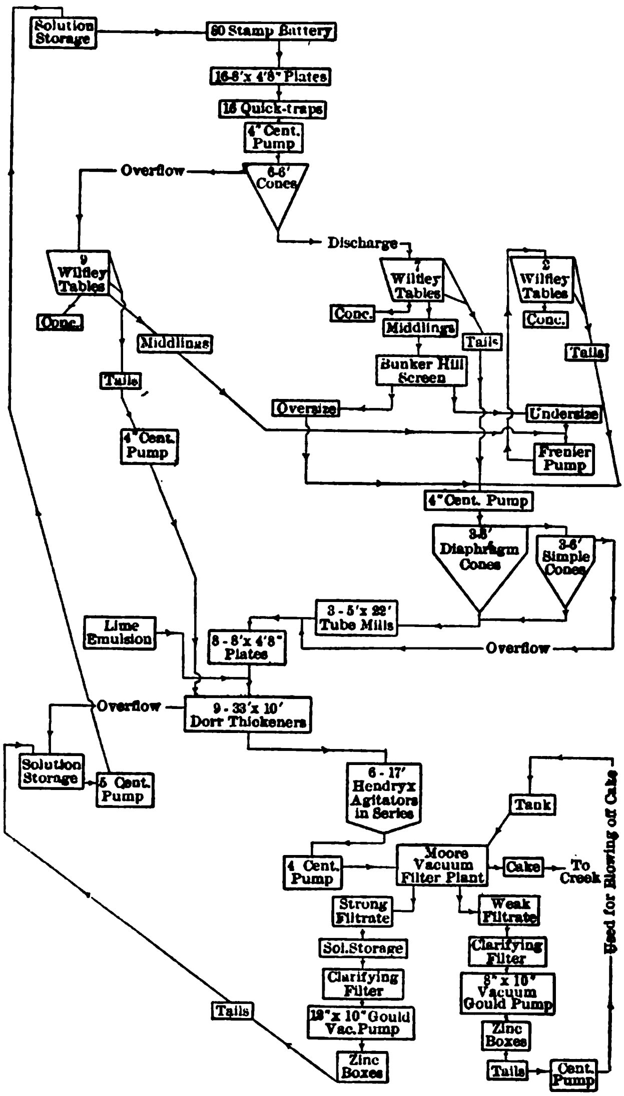
The concentration scheme is only now assuming definite form. Nine Wilfley tables takes the underflow from the Richards classifiers, two take the middling, after removing the coarse on a Bunker Hill screen, and seven take the overflow, after thickening in six 6-ft. cones. The 10 Deister tables are to take the reground sand from the tube-mills. Any oversize tailing from these latter tables is returned to the tube-mills.

This arrangement seems to represent a reasonable economic limit. Further expansion of plant would yield returns on the investment relatively slowly. The great impediment to perfect work is the argillaceous slime. Coagulated by the alkaline solution, it is exceedingly buoyant and sustains coarse material, both sulphide and sand, until dilution is carried to extreme limits. This scheme represents the lesser of two evils. For three years concentration was applied to the tailing, after filtration and dilution with water. An extensive area of canvas with Wilfleys and vanners gave poor returns, and it was evident that the sulphides, probably concentrating to some extent in the tube-mills, were ground so fine as to be irrecoverable. Moreover, what was caught was so high grade as to suggest re-precipitation of silver on the pyrite. The extraction of gold was comparatively good.

That amalgamating is still carried on has occasioned some adverse comment. It is recognized that the absence of this step would materially cheapen and simplify the milling. Sixty stamps, at most, would be required to crush the full tonnage through the coarser screens that could be used. It has been stated that the gold occurs irregularly in the ore, and hence is likely to be coarse. This gold would make an unwelcome element in the concentrate, which has not as yet been made amenable to local treatment. A streak of gold on the tables would be a constant source of danger, and the product would be spotted and difficult to sample for sale. Were it feasible to concentrate successfully after regrinding, the battery plates might well be done away with and the coarse gold allowed to go into the tube-mills with assurance that it would be ground and taken into solution; but that concentration after regrinding is not good practice here seems to have been amply demonstrated. The power charged to concentrating is 30 horse-power.

Regrinding.—The tube-mills are of the Abbé type, tire-mounted and with spiral feed. The tire-mountings were designed by the company's engineers, are amply rigid, and also give complete pro-

tection against possible end-travel of the tires off the rollers. The tires, both on the mill and on the supporting rollers, are of forged steel and promise indefinite life. The drive is by 50-hp. motors, belted to counter-shafts which in turn drive through heavy friction-



FLOW-SHEET, LIBERTY BELL MILL

clutches to the tube-mill shafts. The mills start readily with the use of the clutch, showing a maximum starting-peak of 78 hp.; the running-load varies from 45 to 48 hp. The lining is of 4-in. silex blocks, set in cement mortar, with the narrow edge to the wear. It

gives a year's continuous service. The grinders are 4-in. imported flints, costing $33 per long ton delivered. The ends are lined with local cast iron, and the discharge is through a grating, which will probably give way to the Neal cone-discharge. A typical screen-test is as follows:

Screen Mesh	Feed. Per cent.	Discharge. Per cent.
On 40	47.1	0.7
On 80	29.4	11.4
On 100	8.2	9.2
On 200	6.4	22.3
Through 200	8.9	55.4

As nearly as it is determined, half of the total tonnage crushed is reground. The efficiency of the regrinding varies with the excellence of the separation of the slime. Failure to remove this gives buoyancy to the pulp within and the grinding is poor. With this object in view, the above screen-test is not satisfactory. The practice is to force a diaphragm-cone and to care for the overflow of sand in a simple cone in series. The discharge from the simple cone carries slime out of all proportion. This is mixed with the discharge from the diaphragm-cone to dilute it to 48% moisture. The result is to lose much of the benefit had from the diaphragm-cone. It will undoubtedly prove better to combine the overflow from the three diaphragm-cones in a single simple cone and confine the hampering effect of the fine material to one mill; or, better, a Dorr classifier at this point would be admirable. The feed to the other mills, being then diluted with clear solution, the grinding should be good.

The work of this diaphragm-cone in preparing feed for a tube-mill is remarkable, as the following screen-test shows. The completeness of the elimination of fine suggests the benefits of hindered settling. This is a 6-ft. cone with 60° sides. The diaphragm is 13 in. from the point, with a 1½-in. annular space. The discharge is 1¼ in. diameter.

Screen. Mesh.	Cone Feed. Per cent.	Discharge. Per cent.	Overflow. Per cent.
On 40	27.22	58.95	0.19
On 80	24.82	30.66	8.97
On 100	7.55	5.15	7.44
On 200	8.89	3.42	17.00
Through 200	31.52	1.82	66.40
Moisture	.....	30.80	.....

The absence of results from Dorr classifiers is due to the fact that the two machines installed here were the first, after Mr. Dorr's original installation at Terry. They conformed to his pattern neither in area nor in bottom-slope, being built in what space was available, and were over-loaded. The results have been good through four years' service, but not as good as would have come from the larger machine. The pebbles are fed during the day by the shift-boss, being shoveled into the spiral feed; 135 lb. is the

daily charge. The mills are entirely smooth-running, and the cost of maintenance is at a minimum. The power consumed is 43 hp. per unit.

Dorr Settlers.—The latest step of great importance in improving the mill was the change to continuous from intermittent or charge settling in thickening the pulp for agitation. It is not possible to state definitely that certain measured results followed exclusively from this change. Certain definite improvements were shown by the experimental unit, and large improvements have followed the complete change, but at the same time other conditions were changed. The following are the principal advantages thus obtained: First, continuous extraction is obtained where before the solutions were inactive or re-precipitating; second, a given volume of settler space has 25 or 50% increased capacity, operated on the continuous basis; third, extraction going on in the settlers, the addition of a plant for settling has the added value of supplementing the deficient agitator-volume; and, fourth, labor is reduced one man on each of three 8-hour shifts.

This plant, originally of 5 vats, settling the pulp from the ratio of 5 : 1 to 2.5 : 1, has been increased to 9 vats, settling from the ratio of 9 : 1 to 2 : 1. The increase of solution has come with the addition after the battery of the concentrating-plant, with its great volume of solution for washing and classifying. The four settlers recently installed have been placed out doors, with individual conical roofs and underneath shaft-drive in a conduit, with great saving in cost as against providing the usual mill structure. The power consumed is one-fifth horse-power per unit.

Agitation.—The agitator-capacity was designed with the expectation of using a low-potential electric current to hasten extraction, following the results of extensive experimental work. The plan failed, and, without the current, the provision of space was inadequate. This has been remedied, in large measure, by the addition to the settler-plant, already mentioned. The connecting of all agitators in series for continuous operation was a natural sequence of the adoption of continuous settling. The results appear better, but statistics on which to base exact conclusions as compared with previous charge-agitation are lacking. Some saving in labor of operation and of maintenance is evident. These agitators operate steadily with little attention and very low cost of repairs, but the unit-size is too small for a large-tonnage plant, and the power consumption (6 to 7 hp.) is out of proportion, as compared with Pachuca tank practice or arm-agitation, as carried on at El Oro. No benefit was obtained by spreading the pulp over distributors from the top of the central well, and it is allowed to plunge from the collar of the well. The power consumed is 50 horse-power.

Moore Filter-Plant.—The equalizer, an integral part of the Moore filter-plant, is a simple type of slow-speed agitator (see illustration), efficient for all depths of pulp in the vat and consuming a minimum of power. It has been lately patented and put upon the

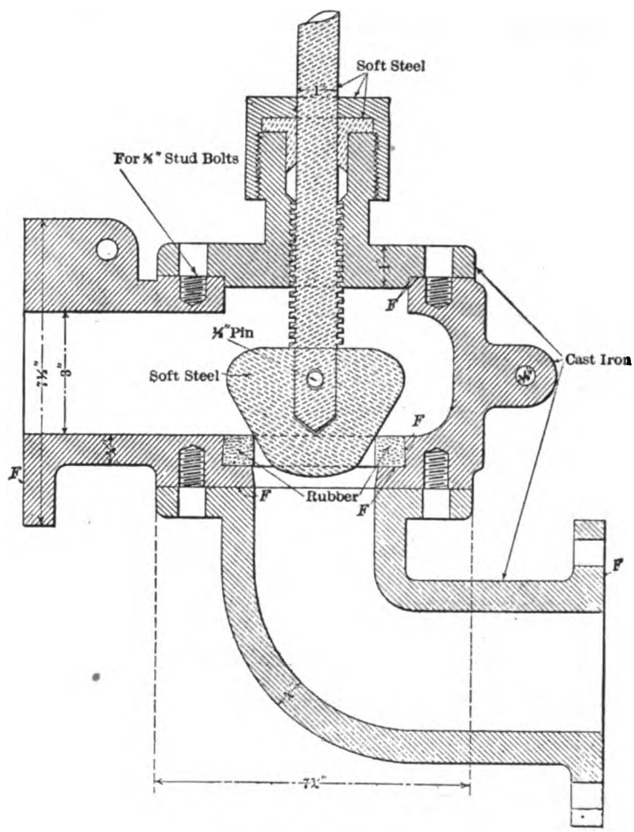
market as the Gordon agitator. The filter-baskets of 66 leaves, each presenting two surfaces 8 by 6.5 ft. of free filtering-surface, are carried on two 10-in. longitudinal I-beams, which are in turn supported by transverse 6-in. beams which extend to the vat walls. The leaves are of No. 6 (20-oz.) canvas, reinforced on both sides at the bottom of the vertical stitching with a strip 3 in. wide of the same canvas. The vertical seams are on 2-in. centres, and the wood strips between are 3/8 by 1/2 in. The frame is of 3/4-in. iron pipe on ends and bottom, and the top is of strap and angle-iron. No cocoa-matting or other filter is used.

The leaves are made at the mill, the sewing (No. 4 linen thread, 1/4-in. stitches) being by power machine (Singer 7-7), and cost complete $12; new canvas alone in place costing $8. The life of a filter is 18 months. All canvas requires hydrochloric acid treatment at intervals of three months. To do this a wash-water vat is brought to 1.25% HCl (18°B.) and to 140°F., the basket is immersed and the liquor circulated with a wet-vacuum pump. All the canvas in use can be treated within 30 hours. The cost of acid is 0.6 cent per ton of ore.

The lifting of baskets is by hydraulic cranes with cylinders 20 in. by 9 ft., the water at 250 lb. pressure. Connection from the crane to water-main is by a specially reinforced coupling on a short length of metallic hose. The raised basket is held by a safety-catch on the crane, and the transfer is effected by a 10-hp., 3-phase, constant-speed motor, with a 3-armed trolley above. Transmission from motor to crane is through a Dodge multiple-disc friction clutch on the motor shaft. The service is severe, but the clutch does well. The vacuum connection from the basket is through a 3-in. hose to a pipe turning in a stuffing-box, and is maintained throughout the transfer.

The greatest single step in perfecting the plant was the change from the common wet-vacuum pumps to a combined dry-vacuum and wet-vacuum system; all entrained air is taken out at the upper end by a dry-vacuum pump, an 8½ by 10-in. vertical duplex air-compressor, and all solution at the lower end with centrifugal pumps in a sump 23 ft. below the top of the filters. Some leaks in canvas will occur, and sand sufficient in quantity to destroy a positive wet-vacuum pump in a few hours is harmless to the centrifugal. Between the dry column at one end and the solution column at the other runs the main for strong solution. Parallel to this and leading from the same dry column runs the weak-solution main, but to a different solution column and pump. Any basket may be connected with either vacuum main, or with the blow-off water-main without disconnecting the hose. The vacuum never fails, being held at 19 to 20 in., near the maximum attainable at the altitude of the mill. To make this possible an excellent valve was designed (see figure). Its appearance is that of a globe valve, and its merit dies in seating an iron cone on a hard-rubber ring of square section. It is quite impossible for sand to lodge on the ring so as to interfere with closing.

Filtration is accomplished in two groups of three vats each, the central one for loading and the two on either side for displacement in water. Basket No. 1 loads in the centre vat and is moved to the wash-water vat at the right. Immediately thereafter basket No. 2 moves from wash-water vat at left to loading vat. The cycle is thus: Loading, 50 min.; transferring and drying, 5 min.; displacing and discharging, 45 to 55 min.; transferring, 5 min. Each load is ¾-in. cake, weighing 2.75 lb. dry per square foot, or 9 tons

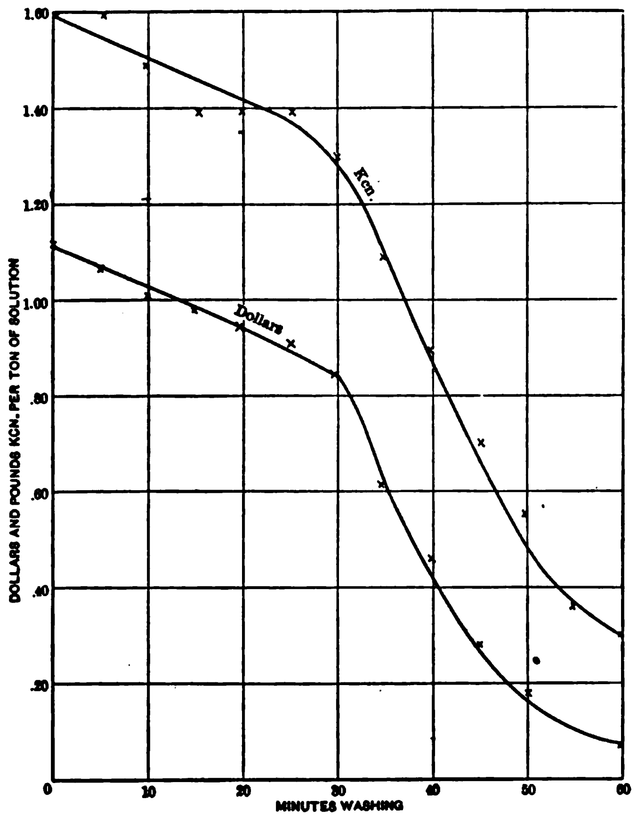


VACUUM-VALVE

per basket-load. This gives a capacity of 108 tons per basket per day, corresponding to 432 tons for the plant. Vertical uniformity of loading is secured by the use of three-air-lifts, which elevate pulp from the bottom of the vats and discharge it over the top.

The practice of displacing at once in water, without an intermediate wash of barren weak solution, can be approved ordinarily only on the assumption of good displacement and low-strength solutions. Displacement is efficient, but the solution (1.6 to 1.75 lb. KCN at this point) is higher than was planned when the plant was

designed. A factor that enters the special problem here is the 8% of moisture brought to the mill in the ore. Average results in washings are shown by the accompanying curve. The cake, partially dried, contains 33% moisture, not taking into account the solution in the pipes and channels, which is difficult to determine, but which must approximate one ton. The cake and passages thus retain a total of 5.5 tons of solution containing 8.8 lb. cyanide and $6.16 in gold and silver. The rate of displacement is 0.15 ton per minute.



WASHING CURVE, MOORE FILTERS

In filtration there are two principal ends to serve: (a) to recover enough strong solution to restore the mill stock; (b) to recover the dissolved gold and silver, and that in solution of such strength as to insure precipitation. Were the ore dry on entering the mill, 36 min. filtration would be sufficient. As the solution drawn in this period contains 7.72 lb. KCN, the efficiency of displacement is 87%, measured in cyanide. As it contains $5.08, the efficiency is 82%, measured in metals. It seems fair to accept 84 or 85% efficiency.

The mechanical loss of cyanide by dilution is that which cannot be restored to the mill stock. The fact that 35 tons of water is brought to the mill with the ore makes it impossible to secure the maximum theoretical efficiency given. From each basket-load the solution recoverable is (5.5–0.8=) 4.7 tons, containing 6.48 lb. KCN. The combined mechanical loss is therefore $\frac{8.8-6.8}{9} = 0.22$ lb. KCN, or 4.7c. per ton of dry ore.

In 55 min. washing, the recovery of gold and silver is $5.93, the apparent loss being 2½c. per ton of dry ore. This seems to be a maximum figure; washing to 70 min. showing almost complete removal. Regular sampling of the solution in washed cakes is not convenient, but, so far as done, it shows 1 to 2c. The loss in cyanide by dilution being so small and the recovery of dissolved metals being complete, the only remaining consideration is the low average strength, 0.75 lb. KCN, of the weak solution. Solution at 0.9 lb. KCN precipitates well. The use of barren wash would insure this strength in the weak solution. On the other hand, 1c. per ton of ore will restore the few tons of the weak solution to strength on the infrequent occasions of poor precipitation. It seems that added costs in depreciation, operation and maintenance would offset any gains from an intermediate wash.

The weak solution after precipitation is used at low pressure, to force the cake from the filter, submerged in wash-water. An advantageous change would probably be to perform this with air and thus return a more nearly dry basket to the loading vat. The removal of tailing is wholly automatic, by reason of the excess of wash-water available. The vat walls run down to three points, across which discharge single jets of water from ¼-in. nozzles, carrying the descending mud through ½ to ¾-in. orifices in the walls opposite the jets.

The operation if four baskets has been described. The fifth is used as a clarifying filter in the seventh vat. All solution, whether decanted or filtered, though apparently clear, requires clarification to insure clean zinc-boxes. In this service the canvas acquires a remarkably fine impervious and tenacious coating. To remove this the basket is returned to pulp filtration after 10 to 14 days. At times a coat of pulp has been gathered on the canvas before using it to clarify, but this seems an unnecessary refinement, which reduces its capacity. The power consumed is 30 horse-power.

The Zinc Precipitation shows little that is unusual. The results are usually excellent with average heads of $1.25 and tailing of 1 to 2c. The flow of solution is 0.7 ton daily per cubic foot of zinc, and 2.4 tons per ton of ore milled, $^3/_{10}$ of the whole mill solution being precipitated. All the solution is metered above the gold-solution vats by a mechanism devised from a common tilting-box tailing-sampler. The pans or pipe guides are always submerged, and, in the manner of dash-pots, steady the movement of

the box. Being placed over the vats, any splash is accounted for in calibrating. Each cycle is registered by a counter. The zinc-lathe is home-built and has a capacity of 700 lb. to 0.001 in. per 8-hr. shift. As cut the zinc is gathered on revolving arms in skeins which fit the boxes. The sludge is gathered semi-monthly, treated with sulphuric acid, washed, dried, and melted. Always high grade, it has recently reached a maximum of 92% bullion. The drying-furnace is of the cast-iron muffle type, and the melting is in No. 150 B. L. crucibles in coke furnaces. The charge for power is 7 horse-power.

Pumping Plant.—The handling of ore and solution necessitates much pumping. This is done with improved Byron Jackson slime-pumps. The pumping units are in duplicates for pulp. The liners used are ¼ to ⅜-in. cast iron. Manganese steel is to be tried. As it stands, the record shows this to be a good centrifugal pump. The life of the liners varies with the thickness of the pulp and the proportion of clay to sand. The tube-mill feed with nearly clean sand gives the shortest life, and the agitator discharge, very thick and with all the clay, gives the longest. Notwithstanding this good service from the centrifugal pumps, I have been for some years of the opinion that the proper pumping equipment for this mill would be a low-pressure compresor with air-lifts for almost all transfers, and to replace the mechanical agitation. The displacement of motors, belting, and shafting, with their need of skilled supervision, would far outweigh the loss of efficiency in the air-lifts. The milling operations would be characterized by extreme simplicity.

Use of Chemicals.—The mill-sheets show this average of the cyanide and protective alkalinity:

	Battery Head.	Second Plate Tailing.	Filter Heads.	Consumption.
KCN	1.75	1.64	1.50	1.48
P. A.	2.32	1.44	1.92	....
CaO	....	....	....	7.5
PbO	....	....	....	0.33

The mixed salt, 99% KCN is used, no advantage being evident at first in trials of the 130% salt, but a recent concession by the makers has led to further trial with the 130% salt, with better results. The figures given are in pounds per ton of solution. Durango lime is used and the figures are in equivalents of caustic soda. Until September, 1909, lead acetate was used, ¼ to $^3/_{10}$ lb. per ton of ore being added at the agitators. Since that time litharge, $^1/_3$ lb. per ton, has been added with the tube-mill feed. An apparent improvement of 5% in silver extraction from the charge is based on inadequate tests.

Cost of Operation.—The operating costs for the period from the beginning of operations with the present type of plant are given on following page. I do not attempt to reproduce more than total department costs. The expense readily chargeable to the various depart-

Year:	1906.	1907.	1908.	1909.	1910.	January. 1911.
Tons per year:	92,900	102,106	116,353	125,681	133,881	149,760
Tons monthly average:	7,742	8,509	9,696	10,473	11,157	12,480
Labor.						
	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.
Superintendent,	1.47	2.03	1.91	1.98	1.85	1.60
Heating,	0.90	1.33	1.34	1.61	1.40	2.48
Electric plant,	1.30	0.76	1.07	1.05	0.76	0.60
Lubricating,	0.54	0.41	0.24	0.32	0.25	0.26
Accidents,	0.54	1.33				
Pumping plant,	......	......	......	......	1.65	1.83
Watchman,	0.17	0.13	0.85	1.20	1.03	1.00
Examination and tests,	......	......	2.72	0.13	0.30	0.54
Total general labor,	4.93	6.00	8.56	6.27	7.24	8.33
Crushing,	2.48	3.16	2.01	4.50	4.80	5.35
Stamping,	18.04	17.28	16.66	14.31	13.70	10.95
Regrinding,	4.94	1.46	0.76	0.66	1.28	1.22
Settling and agitating,	5.20	4.51	4.70	2.75	2.36	2.03
Filtering,	13.65	7.18	6.18	5.24	4.46	4.00
Concentrating,	12.11	11.57	12.27	7.02	5.40	5.05
Amalgamating,	5.71	5.44	4.63	4.42	4.67	5.30
Precipitating,	6.68	4.43	3.42	2.83	2.05	1.96
Total labor,	73.75	61.03	59.56	48.01	46.04	44.19
	Cents.	Cents.	Cents.	Cents.	Cents.	Cents.
Pipe lines,	0.61	1.81	0.52	0.43	1.30	0.25
Bins,	0.89	0.58	0.05	0.37	0.03	
Building,	3.19	2.36	2.72	3.38	3.73	2.79
Electric plant,	2.79	1.44	1.13	0.86	0.48	0.74
Pumping plant,	1.93	1.53	1.01	2.46	1.97	3.13
Heating plant,	4.97	2.80	2.25	2.55	2.58	5.10
Tools,	1.00	0.68	0.22	0.36	0.47	1.02
Cyanide,	40.72	34.19	37.10	31.29	33.90	30.93
Alkali,	5.83	5.91	6.43	5.71	6.45	4.62
Lead salts,	1.22	3.95	4.09	2.85	2.70	4.18
Power,	2.15	3.80	2.96	2.71	2.51	2.24
Light,	1.10	0.42	1.67	1.73	1.65	1.48
Oil and waste,	0.65	1.15	1.00	1.07	0.81	0.86
Assays and melts,	4.17	5.64	4.91	4.00	4.16	4.08
Examinations and tests,	......	1.24	1.59	0.26	0.17	0.07
Miscellaneous,	0.23	0.09	0.06	0.04	0.05	0.05
Total general,	71.37	67.79	67.63	60.24	63.	60.77
Crushing,	4.66	4.80	2.90	2.22	1.77	1.96
Stamping,	17.57	16.85	13.00	13.36	17.80	14.25
Regrinding,	14.89	10.58	8.19	7.05	6.58	6.05
Settling and agitating	4.95	3.00	3.54	5.20	4.34	5.74
Filtering,	13.80	10.98	6.00	5.59	7.06	5.88
Concentrating,	3.69	3.30	2.85	4.59	3.00	1.87
Amalgamating,	4.93	5.53	4.77	3.42	4.04	2.06
Precipitating,	7.75	7.76	6.43	5.68	6.43	5.44
All supplies,	143.72	130.41	115.28	107.35	114.00	104.03
All labor,	73.75	61.03	59.56	48.01	46.	44.19
Total operating costs,	217.47	191.44	174.84	155.36	160.	148.22
Depreciation,	16.	16.	13.	11.	13.	13.Est.
Freight, treatment and discounts,	25.	25.	24.	19.	32	32.Est.
Total metallurgical cost,	258.	232.	212.	185.	205.	193.Est.

OPERATING COSTS, LIBERTY BELL MILL

ments is so disposed and the reason for leaving the other items as a general charge I believe is clear. The general charge for power is the cost of pumping between departments. The first two years were marked by many mechanical difficulties. The benefit of the abandonment of the canvas plant and the change to continuous settling was shown during 1909. The increase in freight, treatment, etc., in 1910 is due to the increased tonnage of concentrate. The balance-sheet and summary of the mill work were given on page 742 of the *Mining and Scientific Press,* May 27, 1911.

These figures are given as a service to the public, following the practice of Arthur Winslow in making public his annual reports on the mine. It is hoped that others may derive from them some return for the benefit that the author and his associates have derived from published accounts, letters, and free access to plants elsewhere. Further, it is hoped that the figures may serve as a warning in some cases and a source of encouragement in others. Certainly, one embarking on a new enterprise under similarly hard conditions can get some measure of the obstacles likely to be encountered. On the other hand, the great improvement of the past two years shows what is possible. This result is primarily the culmination of many years working toward the re-treating system and concentration of operations in mining; a result delayed principally by the harassing labor conditions of 1902-1908, and, in part, by the lack of sufficient early development.

Inasmuch as this mine is situated in a part of the West noted for its high freight-rates, living-costs, and wage-scales (in this case the mine and mill averaging $3.60 and $3.75, respectively, per 8-hr. shift), it furnishes an interesting comparison with the results secured with the alleged 'cheap' labor of Mexico. This comparison is apt, because in its requirements metallurgically the mine is more nearly comparable with the practice at El Oro and Guanajuato than anything north of the line. I wish in closing to acknowledge the courtesy of Mr. Arthur Winslow in granting me permission to publish the many details given, and the valuable assistance of Messrs. W. H. Staver, M. L. Anderson, W. E. Tracy, and H. G. McClain, all of Telluride, Colorado, in collecting drawings and special information.

(August 12, 1911)

The Editor:

Sir—With reference to the description of the Liberty Bell mill, appearing in your issue of June 24, certain statements have received wrong interpretation, so wrong, as to demand correction and explanation.

In saying that the apparent loss was 2.5c. per ton of dry ore, I referred to dissolved metals only, and had no thought that this would be taken as applying to the undissolved value in the ore. In the text, reference is made to page 742, where a table showing

the tailing value is given, and, had this been referred to, there would have been no ambiguity.

In the tabulation of mill costs the word 'Supplies' was omitted in the lower part. This has led to wrong statements as to the costs of a certain step in the process.

Telluride, Colorado, July 26. CHARLES A. CHASE.

MILLING PRACTICE AT THE KOMATA REEFS MINE

By S. D. McMiken

(July 1, 1911)

The Komata Reefs mine is situated on the Hauraki peninsula of the North Island, New Zealand, shown on the accompanying map. When this mine was first worked, dry crushing with stamps, followed by cyanide treatment, was considered the correct method of treating the gold and silver ores of this field. Accordingly, a dry-crushing plant was built, but it was soon found that this method was not suited to the ore and was far too expensive to warrant its continuance. A change to wet crushing, with subsequent treatment of the pulp by cyanide, proved to be advantageous, and all the subsequent improvements in the milling and treatment of ore have been along the line of wet crushing. The extraction during the dry-crushing period was not high compared with the results obtained with the present method of treatment, although the ore milled during the dry-crushing period was worth more than the ore milled during the period covered by the present practice, the ore at that time being 'selected,' to cover the extra expense involved by the dry-crushing process.

Screen Tests of Ore

Percentage on	40 mesh.	60 mesh.	90 mesh.	200 mesh.	Through 200 mesh.
Ore from stamps	38.7	12.0	16.5	14.8	18.0
Ore after mills	nil	1.0	22.0	25.9	51.1
Sand in vats	nil	1.3	23.6	35.0	40.1

Extraction

	Value of tailing.		Percentage.		
	s.	d.	Gold.	Silver.	Value.
Amalgamation on plates			42.0	5.6	38.0
Sand, by cyanidation	2	2	94.2	85.6	93.0
Slime, by cyanidation	0	8	96.5	91.9	95.6
Total, by amalgamation and cyanidation	1	5	97.1	89.2	96.4

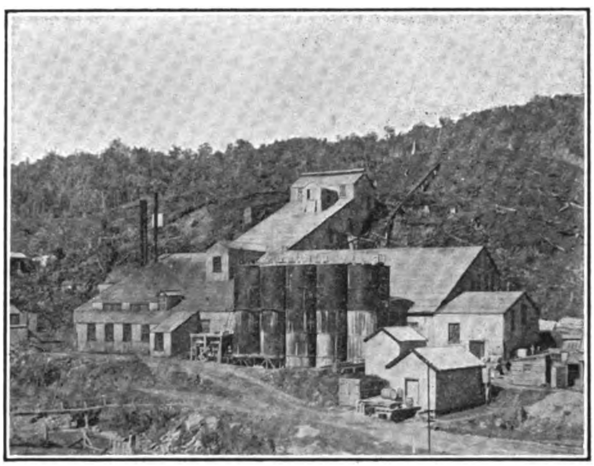
The following definitions are given, for the sake of clearness: 'Slime' is that product, clayey in nature, but with some fine sand, all of which will pass a 200-mesh screen. 'Fine sand' will pass

100 mesh, but remain on 200 mesh. 'Medium sand' is finer than 60 mesh, remaining on 100 mesh. 'Fine' is all pulp, both sand and slime, ground fine enough for cyanide treatment. 'Middle' is sand which has passed through the tube-mills, but is not yet ground fine enough for cyanide treatment. 'Coarse' is the pulp as it issues from the stamp screens, crushed through a woven-wire screen, 4.5 meshes per linear inch. 'Sand residue' is the sand treated by percolation and sluiced into creek, containing a small amount of unrecovered dissolved metal. Cyanide strengths are in terms of potassium cyanide, although sodium cyanide, 130% KCN is used in the plant. The short ton of 2000 lb. is used throughout. The assay value is based on gold at 84s. 9d. per ounce, and silver at 2 shillings.

Nearly all the ore now mined is unoxidized, being obtained from between the 400 and 700-ft. levels. The gold and silver occurs in a quartz gangue, mixed with calcite; in places associated with sulphides of iron and copper, and carrying a proportion of about 5 of silver to 1 of gold. In places, manganese silicate occurs, and native quicksilver has been found. The proportion of sulphides, however, is not great enough to warrant the use of concentration in the mill, and the process is accordingly simplified. The ore does not readily decompose; after six months exposure to the air, it shows only a faint acid reaction, and is in every way amenable to treatment to the cyanide process. The country rock, a highly silicified andesite, is much mixed with the vein material, and as a large quantity of this hard rock has to be crushed along with the softer ore, the duty per stamp is lessened considerably.

The ore from the mine is tipped over a grizzly placed on the main mine hopper. This grizzly is made of feather-edged steel bars, and is 15 ft. long by 3½ wide, the bars being placed 1½ in. apart. The main hopper is divided into two compartments, one for the material that passes through the grizzly, and the other for the coarse lumps of ore. The portion that passes through the bars is about 40% of the total. The two sizes are then trammed to the mill, a distance of three-fourths of a mile, in 4-ton cars. The coarse size is tipped into a hopper situated above the crushers and the small size is tipped directly into the main storage hopper of the battery, where it mixes with the material from the rock-breakers, of which there are two of the Blake type, 12 by 8 in., placed directly above the main storage hopper and crushing to 1½-in. ring. One man looks after both machines, keeping them fed with ore, as well as attending to the friction hoist used for handling the four-ton cars up a small incline to the top of the mill. The ore is then fed by automatic feeders to four batteries of five stamps each, crushing in water through a steel woven-wire screen of 4½ holes to the linear inch. From the batteries, the coarse pulp flows to two tube-mills, thence to an elevator-wheel which raises the pulp to a separating-box where the fine overflows and passes to the amalgamating plates, and the middle is returned to the tube-mills for further grinding. From the plates, the fine passes to

another elevator-wheel which raises the pulp to the spitzkasten, where the sand is separated from the slime. The sand is run into the percolation vats; which are furnished with radial distributors of special design, and after treatment by cyanide solution, is discharged into a creek near the plant, the gold-bearing solution being led to precipitation-boxes filled with zinc-shavings, and then into the usual storage sumps. The slime from the spitzkasten is run into a settling vat provided with a continuous discharge at the bottom, limewater being added, and is there thickened to about 40% water, and then pumped into Brown & McMiken tanks (Pachuca) where cyanide solution is added, and agitation maintained until the gold and silver has practically all gone into solution. The slime is then pumped into another settling vat similar in design to the one already mentioned, and here another thickening

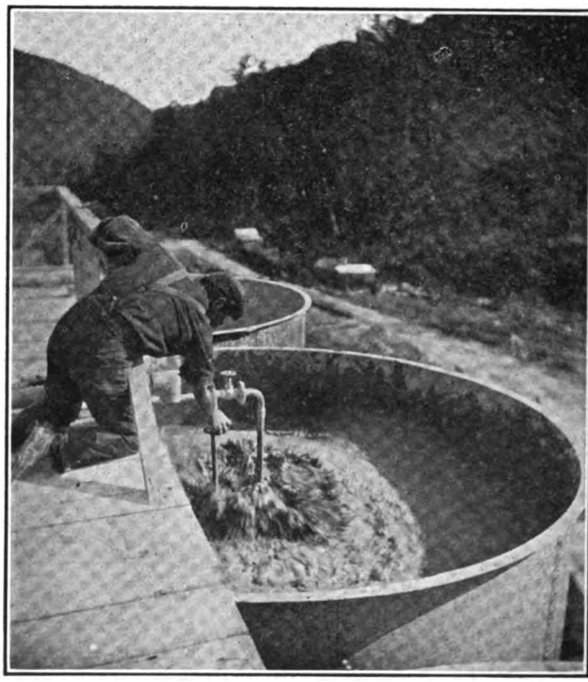


KOMATA REEFS MILL

takes place, reducing the percentage of moisture to about 45. The solution which is decanted off the settler is run into a vat provided with a sand filter-bottom, and thence to the precipitation-boxes. The thickened slime is then pumped from the discharge of the settler to a Moore filter of special design, where the slime is formed into cakes and washed thoroughly, the residue being finally sluiced into the creek. The precipitate from the zinc-boxes is removed during the monthly clean-up, and is treated with sulphuric acid, washed, dried, and fluxed with borax glass in plumbago crucibles, heated in gasoline furnaces of patent design. The resulting bullion is shipped to London and sold at market rates. The amalgam from the plates is retorted, melted, and sold to the local bank in preference to shipping it.

The crushing machinery consists of 20 stamps arranged in four batteries of five stamps each, ten stamps to each cam-shaft.

The batteries are set on Kauri wood blocks of ample size, which rest on a concrete foundation, giving a firm and satisfactory crushing base. The frame is of the front-knee pattern, strongly braced, and giving little vibration. Each mortar-box weighs three tons and is of the single-discharge type, fitted with back and end liners of hard cast iron. The boxes are bolted down to the wooden blocks with eight 2-in. bolts, a single sheet of rubber packing ¼-in. thick being placed between the wood and the mortar. The weight of a stamp complete with new shoe is, stem, 480 lb.; head, 380 lb.; tappet, 130 lb.; shoe, 170 lb.; total, 1100 lb. The average working



B. & M. TANKS AT KOMATA REEFS

weight is about 1030 lb. A system of using the shoes as dies, when half worn out, has been in vogue for over eight years and has effected a great saving. No dies are ever bought, the half-worn shoe being turned upside down and the neck placed in a special casting in the bottom of the mortar-box. This allows the shoe to be worn down to within ½-in. of the neck, when it is pried out and another shoe inserted. Solid cast iron guides are used, and have given no trouble for years. They are preferable to wooden or babbited guides. The tappets are of the 3-key type and seldom slip. Finger props are used to hang up the stamps when required. The usual drop is 6½ in., and the speed is 100

drops per minute, the ore being fed into the mortar by means of an improved Challenge feeder of the rotary table type, a special pawl motion being used in preference to the ordinary 'frying-pan' variety. Both cam-shafts are 6 in. diam. and each carry ten cams of the Blanton type. These cams are preferable to the old double-keyed kind, which give much trouble when they get loose on the shaft. The cam-pulley is also fitted with the Blanton wedge-piece, and never gets loose. Various kinds of screens have been tried, including punched and slotted screens; after exhaustive tests, the steel woven-wire screen has been found to give the best results as regards efficiency and cost. The kind now used is made to order in America, and is very good, a set of screens having lasted nine weeks. The ore is crushed through a woven-wire screen of 4½ holes to the linear inch, the thickness of the wire being 20-gauge. The discharge is kept about 4 in. above the dies, and about ten tons of water per ton of ore is used in crushing. The duty per stamp is five tons per day of 24 hours, and the horse-power used 48, including friction. Exhaustive trials of various makes of shoes, steel and iron, have been made, and the best, from a commercial point of view, is a shoe made of a special mixture of hard white iron manufactured by a local foundry. The consumption of iron, in shoes, per ton of ore crushed is 0.67 lb. The coarse pulp from the stamps is delivered into a steel launder running the full length of batteries, and immediately in front of the mortar-boxes. This launder is set with a grade of 1 in 12 in order to conduct the coarse product of the stamps to the tube-mills, at a lower level. This launder is divided into two sections just before reaching the tube-mills, so that each mill gets an equal quantity of the pulp to grind (see grade of pulp in tables). The power used by the battery is furnished by a 6-ft. Pelton wheel, working under a head of 170 ft., and two single-cylinder condensing engines, supplied by two locomotive-type boilers, coal-fired. In winter, the water-power amounts to about 80 hp., but in summer the whole battery is run by steam. It requires about 140 hp. to run the whole plant.

Grinding is accomplished in two tube-mills, one of which is 16 ft. long by 4 ft. diam., and the other is 15 ft. long by 4 ft. 4 in. diam. The former was the first steel-shell tube-mill made in New Zealand for grinding battery pulp. It is furnished with spur gearing and runs on trunnions. The second mill was made out of a 'White' rotary dryer, cast-iron shell, running on trunnions. At first, this mill was run on friction rollers, but as these proved unsatisfactory, end trunnions were substituted, and there has been no trouble since. Both of these mills are lined with Brown's patent liner, which consists of a liner of hard cast iron, with ribs bolted over it; they last 22 months, grinding 28,500 tons of hard quartz, during the life of a set. The consumption of iron is 0.48 lb. per ton of ore ground to a finished product, 50% of which will pass a 200-mesh screen. This lining has been adopted by most of the mining companies in this field, in preference to all others, and will

probably supersede all others in countries where fine grinding in tube-mills is in vogue.

Each mill is fed from a pointed box, 3 ft. deep and 3 ft. square at top, by means of a pipe 1½ in. diam. projecting into the trunnion of the mill. The size of this box is so adjusted that a large proportion of the water together with some slime is allowed to run off the top into a launder, so that the product issuing from the pipe is about 1¼ parts water to 1 part ore, by weight. It has been found that this proportion is the best for grinding our ore in tube-mills. The 16 by 4-ft. mill revolves at 29 revolutions per minute, and the 15 by 4 ft. 4 in. at 28 revolutions, according to the formula $R=\frac{200}{\sqrt{D}}$, where R equals revolutions per minute and D equals interior diameter of mill in inches. This speed has been found to give the best results, and the formula is recommended to users of tube-mills. The flints used are French flints about the size of a hen's egg. Larger and smaller flints have been tried, but this size gives the best results. Selected pieces of hard quartz were also tried but proved unsuitable. The charge used in the mills is such that the level of the pebbles is just even with the lower edge of the discharge orifice, which in this case is 8 in. diam. and is furnished with a screen of three holes per linear inch, in order to prevent the flints from escaping with the ground material. The wear of flints per ton crushed through the stamps is 2.2 lb. Each mill takes 25 hp. when running at the above speed, and there are practically no repairs, other than renewing the lining and replenishing the supply of flints. Each mill is furnished with a Heywood & Bridge friction clutch, which is much better than fast-and-loose pulleys. At this battery, these two mills have been running for years, and the trunnion bearings are as good as when first put in, the original babbitt metal being still there. The whole of the floor under the tube-mills is finished in concrete, and a gutter leads into a small sump in order that any spilt sand may be picked up. From the discharge end of the tube-mills a steel-lined wooden launder conveys the middle, fine, and overflow from the tube-mill feed-boxes to an elevator-wheel, 18 ft. diam., which lifts the pulp to a pointed box furnished with a 3-in. discharge cock at the bottom, and an upward current of water. This box serves to separate the material that is ground fine enough for treatment, from the coarser product, which issues from the 3-in. discharge and is returned to the tube-mills for further grinding. The product which is too coarse for treatment is thus returned again and again to the mills until ground fine enough for treatment by the cyanide process. The elevator-wheel is built of wood, and iron and the hub and buckets of iron. This wheel works well and has given no trouble since it was built, five years ago. (See grade of pulp after mills.)

From the overflow of the separating-box, the fine pulp flows in a wide launder to the amalgamating room. During its passage

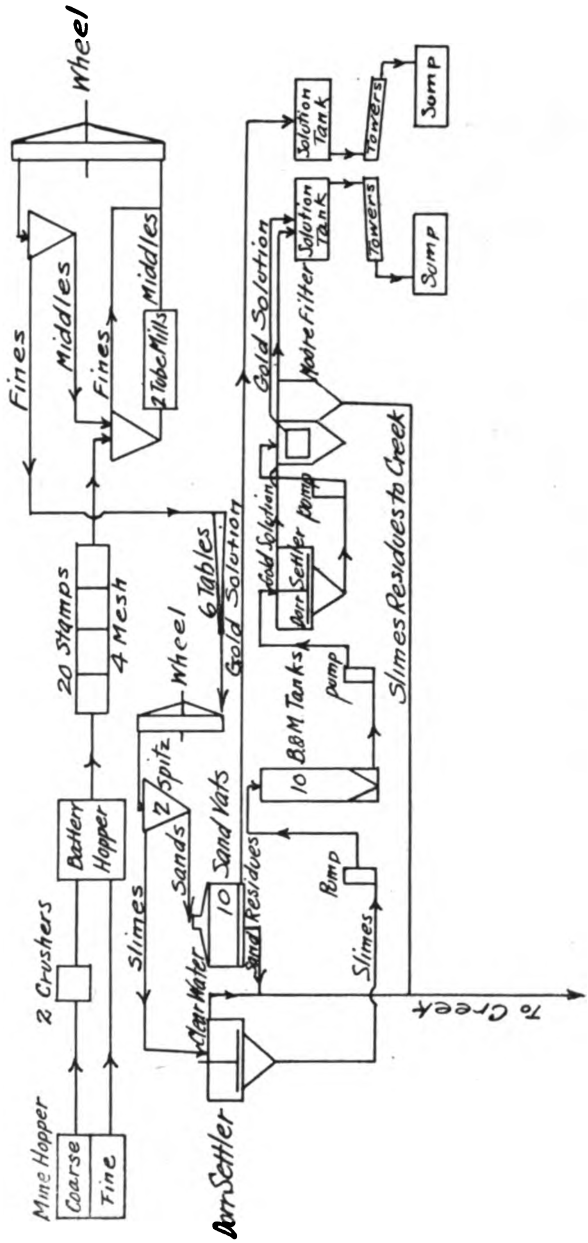
along this launder the pulp is automatically sampled in duplicate, and these samples taken daily for assay. The amalgamating room is situated some distance from the stamps and is built upon the solid ground, hence there is an entire absence of vibration. The floor is cemented throughout, and is given a fall to one side, 1 ft. in 15 ft. This allows all spilt material to run into the main pulp launder from the tables, the launder being situated underneath the edge of the floor. There are six amalgamating plates of muntz metal, 10 by 5 ft.; copper plates were tried years ago, but were afterward replaced by plates of muntz metal, which are more easily kept in order and save a higher percentage. There is practically no absorption of gold, and consequently very little value is locked up in the plates. A little dilute sulphuric acid is rubbed on the plates now and again; outside of this nothing is done to them beyond adding mercury as required. A device for turning the flow of pulp off the plates, one at a time, when cleaning up, is used, and the diverted pulp is returned to the elevator-wheel. There are two riffles at the bottom of each plate, as well as a mercury trap to each, and finally there is a large mercury trap in the main launder carrying the pulp away from the plates to the second elevator-wheel. The recovery by amalgamation is 38% of the total value. The plate room is under lock and key, and is only entered by the amalgamator when necessary.

The pulp from the tables is lifted by a wheel, similar to the one described, to a height of 16 ft., and delivered into a steel spitzkasten, furnished with an upward current of water, and a 2-in. discharge cock at the bottom. The spitzkasten is 4 ft. square and the sides slope at an angle of 60°; the overflow passes to a second one, 6 ft. square with similar sloping sides, also provided with a 2-in. discharge and upward current of water. This water supply is kept at a constant head of 40 ft. above the 'spitz.' This arrangement has been found to give the best results for separating the sand from the slime. The product issuing from the bottom of each 'spitz' is allowed to mix in a common launder and flow to the sand-treatment plant, while the overflow from the second 'spitz' goes to the slime-treatment plant.

All the separated sand is conveyed to the sand vats, nine in number. These are built of steel, and are 22 ft. diam. by 8 ft. deep. Each of these vats is fitted with a filter-bottom, consisting of half-round slats of wood, 1 in. wide and 1 in. apart, supported on 4 by 1-in. wooden laths on their edge, and over the slats is stretched a filter-cloth of ordinary jute sacking. This cloth is firmly fixed all round the edge of the vat by means of ½-in. diam. rope, caulked into a groove. These cloths last about three years, are very cheap, and easily put in place. Each vat is furnished with a distributor consisting of a wooden launder divided into four divisions, discharging at different peripheral points, the feed being at the centre, and the whole apparatus revolving on a central spindle by the force of the sand and water leaving the curved ends of the divisions. This distributor fills the vat evenly, little

shoveling being required to level the top of the charge, previous to treatment by cyanide. The surplus water is taken away from the centre of the vat and conveyed to a pointed box, 12 ft. square and 12 ft. deep, where lime-water is added, and the fine sand which has escaped from the vat is settled and pumped back to the launder which conveys the slime to the slime-treatment plant. The usual charge for a vat is 80 tons of sand, the grade of which is given in tables. The sand is allowed to drain as dry as possible, assisted by a small vacuum-pump, if necessary, and is then sampled by boring holes with a special sampling auger, the holes being bored to a depth of 6 ft. This sample is taken in duplicate, and assayed in duplicate, and the result recorded. The amount of moisture left in the sand before treatment by cyanide is usually about 14 to 15 per cent.

A wash of 20 tons of cyanide solution of 0.05% KCN is then run on top of the charge, and percolation started. This wash serves to pave the way for the strong solution, and it is percolated rapidly through the charge, passed through a zinc precipitating-box, and into a weak sump, where the solution is sampled, and if the assays show no value, the solution is thrown away, as it is too weak to be of any further use. The first wash dissolves very little of the gold and silver, but is useful in reducing the consumption of cyanide when the strong solution is put on. The first wash is allowed to drain off, and then 20 tons of solution at 0.08% is run on the charge and percolation continued. As soon as the whole charge is saturated with the strong solution, and there is still about four inches of solution on top of the charge, percolation to the precipitation-boxes is stopped, and the solution is pumped back to the top of the charge by means of an adaption of the well known air-lift, one to each vat. The strong solution is thus kept circulating for eight days without leaving the vat, and is thoroughly aerated at the same time. Caustic soda is added to the strong solution, in order to provide sufficient alkalinity (Clennell's method of estimation being used), and a considerable saving in cyanide is effected by its use. The strong solution is then allowed to percolate from the vat into the strong-solution zinc-boxes, and thence into the strong-solution storage-sump. When the strong solution leaves the top of the charge dry, weak solution of 0.08% is pumped on, and about 200 tons of this weak wash is passed through the charge, the weak solution being passed to the weak-solution zinc-boxes, of which there are two, and thence into the weak-solution storage-sump. Finally, a water-wash of about 10 tons is put on the charge, and percolated through a separate wash-solution zinc-box, into a last wash-sump. The time taken by the washes is generally about 14 days, and after all the final wash has drained off, a door in the bottom of the vat is opened and the whole charge of sand is sluiced out, by water under a pressure of 65 lb. per square inch, into the creek. The cloth is swept clean with a coarse broom, the door put on again, and the vat filled with water ready for a fresh charge. It has



FLOW-SHEET OF KOMATA REEFS MILL

been found beneficial to fill the vat with water, so that the overflow takes place as soon as filling starts. This prevents an undue settlement of fine material on the cloth, which, if allowed, would hinder the rate of percolation.

The slime and fine sand passing 200 mesh, from the overflow of the second spitzkasten, is conveyed by launder to a mechanically operated settler provided with overflow launder at top. This settler consists of a steel vat 22 ft. diam. by 8 ft. deep, with conical bottom. Lime-water is added to the slime as it passes along the launder to this settler, and the slime settles very rapidly, the overflow being quite clear. The lime is fed automatically into a mixer at the average rate of 20 lb. per ton of dry slime, but there is a controlling device whereby from 1 to 50 lb. per ton can be fed as required. The thickened slime flows from the bottom discharge of the settler continuously, and contains an average moisture content of 40% water. This thick slime is piped a short distance to a double-plunger outside-packed pump, each plunger of which is 6-in. diam. by 15-in. stroke, geared, with cast-iron mushroom valves. The wear and tear on valves and plunger is not great, the plungers being renewed every eight or nine months, and the valves every four or five months. By means of this pipe the slime is elevated through a 3-in. pipe to the Brown & McMiken patent air-agitation tanks (Pachuca), of which there are ten, each 38 ft. high by 10 ft. diam., built of steel, with a cone of 55° pitch, and furnished with the usual air-lift and pipes. These agitators were first used at this (Komata Reefs) mill, but their use has extended all over the metallurgical world, as they are the most economical and efficient agitators yet invented. The cost of upkeep of these agitators is very low, as they only require a new 'nozzle' valve on the pipes now and then, and only 8 hp. is required for the whole ten, using compressed air at 22 lb. per square inch. No trouble is experienced in keeping the charge agitated while filling or emptying the tanks, and there is no accumulation of heavy material on the conical bottom. The slime is diluted by adding cyanide solution in such quantity that the total solution and moisture represents 65% of the whole. Each tank holds 40 tons of dry slime. The cyanide solution is put on at 0.1% strength, and agitation is kept up for seven or eight days, when all the metals are in solution. The slime is then pumped by another plunger pump to a second settler. Here the slime is thickened to 45% moisture, the cyanide solution continuously decanting into a sand filter. This sand filter is a steel vat, 22 ft. diam. by 4 ft. deep, furnished with a lath bottom supporting hessian cloth, which is covered with a layer of battery sand, about 4 in. deep. The solution percolates through this layer of sand, which clarifies it, and passes to the slime-solution zinc-boxes, of which there are two, and thence into the slime-solution sump. The thickened slime from the second settler is then pumped by means of a plunger-pump into the slime-tank of a modified type of Moore filter. This type, termed the turnover filter, was first used in this mill, being

designed by me in 1906. It has proved to be economical in working cost as well as being cheap to install. The filter consists of ten cells, each 7 ft. square, formed by placing sheets of V-shaped corrugated iron inside a frame of 6 by 3-in. timbers, grooved on the inside to receive the edges and ends of the corrugated iron, the whole being covered with 8-oz. duck, fastened to the frame by means of slats nailed over the duck. A corner of each frame is bolted to a Y-shaped arm of cast iron, which is keyed to a shaft which forms the pivot on which the filter turns over.

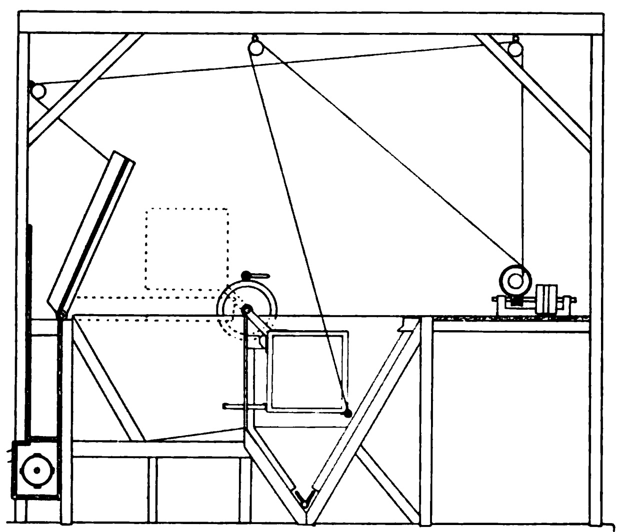
The inside of each cell is connected to a 2-in. header pipe by means of a flexible piece of suction hose and union joint, the 2-in. header pipe being in turn connected, by a piece of suction hose, to the pipe leading to the vacuum pump. The filter contains an area of 200 sq. ft. of filtering surface, and is capable of treating efficiently 50 tons per day of 24 hours. The filter is lifted by a worm-geared hoist which allows the filter to be suspended in any position required, the discharging platform being lifted by a second drum on the same hoist. The washed slime-cakes are dropped into a mixer, where water is added and the whole mixed up into a slime again and run into the creek. In practice, a cake 1¼ in. thick is built up in about 30 min., under a vacuum of 24 to 26 in., produced by a Tangye belt-driven pump, 6 in. diam. by 12-in. stroke, which delivers the solution into another sand-filter, similar to the one already described, and thence into the zinc-boxes. When the filter cells are immersed in the wash-tank, the solution is pumped direct to the wash-solution zinc-boxes, and flows back again to the wash-tank, thus maintaining an even quantity of solution in the wash-tank. This wash solution is generally about 0.03% strength. The cakes are dried as much as possible before being discharged, and they retain 25% moisture. The cycle of operations, namely, forming cake, washing, and discharging, takes 1¼ hours.

The precipitation-room is 42 ft. long by 22 ft. wide, and the windows are all securely barred; one wall is fitted with openings, so that the interior of the room is always under observation. The floor is concrete, finished with neat cement, and provided with gutters to convey any spilt material to a small sump. In one corner, a reinforced concrete strong-room contains a large Milner safe, for the safe-keeping of amalgam and bullion.

The zinc precipitation-boxes, nine in number, are all the same size, namely, 14 ft. long, divided into ten compartments, 2 by 2 by 1½ ft., fitted with 4-mesh screen false bottoms, to allow the precipitate to accumulate between the screen and the bottom compartment. Each compartment is fitted with a 2-in. wooden plug inserted in a hole in one corner to facilitate cleaning up, and each box has a launder, running the full length of box, to convey the precipitate to a clean-up vat, 4 ft. square by 2 ft. deep. This vat is furnished with a filter-cloth on the bottom, and the space beneath the cloth is connected by a pipe to a steel receiver 10 by 3 ft., which is again connected to a small vacuum pump, 6 in. diam

by 9-in. stroke. The solution from the sand and slime treatment is run through zinc-boxes, and into the various sumps, of which there are five, each capable of holding 40 tons of solution. The zinc shaving used is cut at a local foundry, and is dipped in a solution of acetate of lead before being placed in the boxes. Acetate of lead has been found to improve the precipitation when dealing with weak and foul solutions, and, as a rule, the solutions leaving the zinc-boxes rarely assay more than 4d. per ton.

A clean-up is made every four weeks, and the method is as follows: The solution flowing through a box having been shut off, the coarse zinc is shaken and washed in the first, or top cell, and then taken out and placed on a tray. The plug in the bottom of the cell is then removed and the broken zinc, slime, and solution,



TURN-OVER VACUUM FILTER, KOMATA REEFS

are run into the clean-up vat, the cell is then sluiced out with a few quarts of solution from one of the other cells, and the plug replaced. The coarse zinc on the tray is replaced in the cell just emptied, and the coarse zinc in the next cell washed and placed with it. The same procedure is adopted with each of the cells in the box, and when all are replenished with zinc shaving, the solution is turned on again, and precipitation proceeds as before. All the time that the slime and short-zinc are running into the clean-up vat a vacuum-pump is taking away the solution and leaving the mixture of short-zinc and bullion slime on the filter-cloth. When fairly dry, a water-wash is put on the filter, and when again dry, the mixture is shoveled into a circular wooden vat, 6 ft. diam. by 2 ft. deep, furnished with stirring gear. A weak solution of

sulphuric acid, 10 parts water to 1 part acid, is run on the slime, and the whole stirred until all the short-zinc is dissolved. The whole of the content of the acid vat is then transferred to a second vacuum-filter box, where several water-washes are added, and all the zinc sulphate washed out of the precipitate. The slime is then placed on thin iron trays, 3 ft. square, and taken to the melting-room.

The melting-room is detached from the main mill building, as the insurance companies do not permit the use of gasoline as a fuel inside insured premises. It is a room 30 by 20 ft., built of iron and wood, with a concrete floor, and ample provision is made for carrying off the gases. The trays of precipitate from the vacuum-filter are placed in a drying-furnace made of sheet iron. This furnace is 6 ft. high by 3½ ft. square, and has three openings for shelves for receiving the trays of precipitate; the fire is placed on a grate near the bottom of the furnace, and the flame travels in a zig-zag manner under and over the shelves and escapes into a flue at the top of the furnace. This furnace will dry over 800 lb. of precipitate in 8 hours, wood being used as fuel.

The dried slime is then transferred to a rotary tubular screen, half filled with flints, and there reduced to a fine powder and mixed with borax glass in the proportion of 5 parts slime to 1 part borax glass. The mixture is melted down in plumbago crucibles (Morgan's No. 70), the pots being placed in specially designed furnaces, benzine being used as fuel. The benzine is stored in a tank of 50 cu. ft. capacity, placed on the hillside 70 ft. above the melting room, and conveyed to the melting room by means of ½-in. diam. galvanized pipe. Four furnaces of special shape, designed by me, are used for smelting the slime and the resulting bullion is over 900 fine, the proportion of gold to silver being generally about 1 to 8. The bars of bullion are dipped in powdered nitre in order to clean the outside of the bars, and are then polished with a scratch-brush, sampled, and sent to the local bank for shipment. The slag is very clean and is crushed by a single stamp to recover any 'shot' of bullion that may be retained. The crushed slag is re-melted periodically, and the final slag rarely assays more than £4 per ton. The amalgam from the plates is retorted in a horizontal retort, fitted with trays, and the retorted metal is melted down and cast into bars of suitable size and sold to the local bank. The fineness of the bars is about 900, the gold fineness being 600 and the silver 300. The cost of melting is 1.34d. per ounce.

The assay plant is situated a short distance from the mill; the building consists of a sampling room, furnace room, balance room, chemical laboratory, and store-room. The sampling room is furnished with the usual sampling and quartering apparatus, a drying oven is used for drying the samples and evaporating solutions, and there is a sample crusher and grinder driven by a small Pelton wheel. Gasoline furnaces are used, both for fluxing and cupelling. These are made on the premises, and are of better shape than the imported furnaces; they are made by mixing broken

crucibles and fire-clay together and ramming into a thin iron frame. After being allowed to dry for some days, they are finally baked into a hard brick. Gasoline under a head of 70 ft., is used in 'Cary' burners; a 1½-in. burner is used for the fluxing furnace. The chemical room is fitted with all the appliances for analysis, testing of ores by cyanide, and other processes, electrolytic experiments, etc., and has in addition to these, a small tube-mill, and four air-agitation tanks, each 18 in. high by 5 in. diam. There is also a small filter in conjunction with this plant, and we are thus enabled to make tests on ore on exactly the same lines as in a working plant.

NORTH WASHINGTON POWER & REDUCTION MILL

By M. H. Joseph

(July 8, 1911)

The mines at Republic have previously had but a small and precarious outlet for their ores, dependent on the demands of distant smelters for highly silicious materials, especially for use in converter linings. Of late the smelters have used the Republic ores in fluxing the lead ores of the Couer d'Alene and other districts, but as no ore containing less than $11 in gold and silver per ton will pay a profit over the cost of mining, transportation, and treatment at distant reduction centres, a less expensive treatment for lower grades of ore by local milling processes has become a necessity. In the milling practice formerly in vogue in Republic the mechanical limitations of fine grinding and the subsequent treatment of finely ground pulp, caused great difficulty. The gold in the ores of the camp, though mechanically free in a dense matrix, is ordinarily so finely divided as to be invisible, except by microscopic examination. Silver occurs in some of the high-grade ores in the form of selenide of silver. Native silver is occasionally found in medium-grade ore, and it occurs throughout the ore as a sub-sulphide of silver and also alloyed with gold. Various analyses and metallurgical tests have proved that the incomplete recovery of gold and silver by cyanidation at Republic is due, not to the presence of rebellious minerals, but to the density of the ore which would not permit permeation by the cyanide solution.

The new mill, designed by H. W. Newton, will treat 1000 tons of ore per day and will be constructed in eight units of 125 tons each, two of which were expected to begin operation about June 1. In addition to the main building, there will be a crushing plant, refinery, assay office, pumping station, warehouse, and business office. The machinery will be electric-driven with power generated at the company's plant, at Similkameen Falls, near Oroville, Washington, to supply power and light for the mines and mill, and to light the town of Republic. An auxuliary plant will be included in the mill equipment, with 325 kw. of electric power available for use, in case of interruption of power along the transmission line.

A spur of the Spokane Falls & Northern railway crosses the millsite, and over it the ore will be conveyed from the mines to the storage-bins of the crusher-plant, from which it will go through the following stages of treatment: From the storage-bins the ore is fed to Blake crushers by a shaking-screen, the undersize, together with the product from the crushers, falling to grizzly bars spaced to one inch. The oversize from the grizzlies is fed to Dodge crushers, and the product, joining the grizzly oversize, gravitates to trommels. In the trommels, all the material passing a ¼-in. ring is removed, the oversize passing to rolls set to crush ¼ in. The product of the rolls, with the undersize from the screens, is carried by a conveyor-belt to the mill-bins, where an automatic sampler removes a cut for assay, the reject falling to a second conveyor-belt, which automatically distributes the ore to the bins.

From the mill-bins the ore is fed by plunger feeders to Trent Chilean mills, where it is ground in cyanide solution to pass 20-mesh screens. The product of these mills flows to Akins classifiers, where all the material fine enough for cyanide treatment overflows to Trent agitators, the classified sands dropping to the feed-box of the tube-mill, where these sands are reground so that 85% will pass 200 mesh, after which they flow to the elevator sumps. From the elevator sumps bucket-elevators return the pulp to the Akins classifiers, where any remaining unfinished product is removed, the finished pulp going directly to the Trent agitators, connected in series, where the pulp is agitated, passing through the series of tanks by displacement to disc-thickeners.

In the disc-thickeners the pulp is relieved of a portion of its enriched solution, which passes to the precipitation department, the thickened pulp flowing to Oliver filters, which separate the remaining solution from the ore by vacuum filtration, the solution passing to the precipitation department and the ore to the tailing dump. In the precipitation department the solution is mixed with the quantity of zinc dust necessary for the precipitation of the gold and silver, and is pumped to the refinery, where the gold and silver precipitate is removed in a filter-press, the barren solution again entering the mill circulation. The precipitate is periodically removed from the press, melted, and cast into bars.

MINE AND MILL OF THE BRAKPAN MINES, LTD.

By H. S. Gieser and C. A. Tupper

(December 23, 1911)

In this article the description of the mine is due to H. S. Gieser, while C. A. Tupper has contributed to the discription of the mill. The property of the Brakpan Mines Ltd. is situated on the far east Rand, 21 miles from Johannesburg. It consists of deep-level claims opened by two 7-compartment shafts 4400 ft. apart in the direction of the dip. The No. 1, or north shaft, is 3100 ft. deep vertically: No. 2, or the south shaft, is 3700 ft. deep, at which point the reef

was intersected. An incline with an average grade of 7½° connects the two, and from this six levels are driven in both directions. Underground haulage in the incline is effected by means of an endless rope.

The electric power is purchased from the Victoria Falls & Transvaal Power Co., and is used throughout the plant, over 10,000 hp. being taken, distributed as follows: Main hoisting-engines, 4500 hp.; mine pumps, 1500 hp.; battery and tube-mills, 1950 hp.; reduction works pumping plant, 1000 hp. The remainder is used on the surface and for underground ventilation and haulage. All transmitting cables are paper-insulated, lead-covered, and armored, and are laid underground throughout, upward of 30,000 ft. having been placed. To a large extent they are laid on the ring system, thus preventing serious stoppage due to cable failure or damage. Power is metered to show the consumption for pumping, hoisting, milling, etc.; also, each pair of feeders is separately metered, enabling the supply to different departments to be closely checked.

The No. 2 shaft has 7 compartments, measuring 42 by 9 ft. over all. It is provided with a steel head-frame and is served by 2 electric hoists. One has parallel drums 11 ft. diameter and 8 ft. wide, and is grooved for 1½-in. rope. Tooth-clutches are provided for adjustment, also the standard type of liquid starter and automatic overhoisting prevention devices. The post brakes are applied by a dead weight relieved by air-operated cylinders with the ordinary cataract cylinder attachments. The second hoist, the main dimensions of which are the same as the other, is of the Ilgner system, with Ward-Leonard control, direct current, the motor generating-set being placed in a subdivision of the engine-room.

No. 1 shaft is equipped with a steel head-frame and a Ward-Leonard hoist similar to those erected at No. 2 shaft. All three hoists are 1500 hp. each and have a capacity of 100 tons per hour from a vertical depth of 3800 ft. The maximum hoisting speed is 3500 ft. per minute, and the average time for a complete trip is about two minutes.

The reduction plant has recently started operation, with 2,000,000 tons of 6.62 dwt. ore in reserve in the mine, where there is a stoping width of 55 in. Aside from pneumatic slime-agitation in Pachuca tanks, followed by vacuum filtration, with zinc dust precipitation and subsequent filtration in Merrill presses, the features of interest include the following:

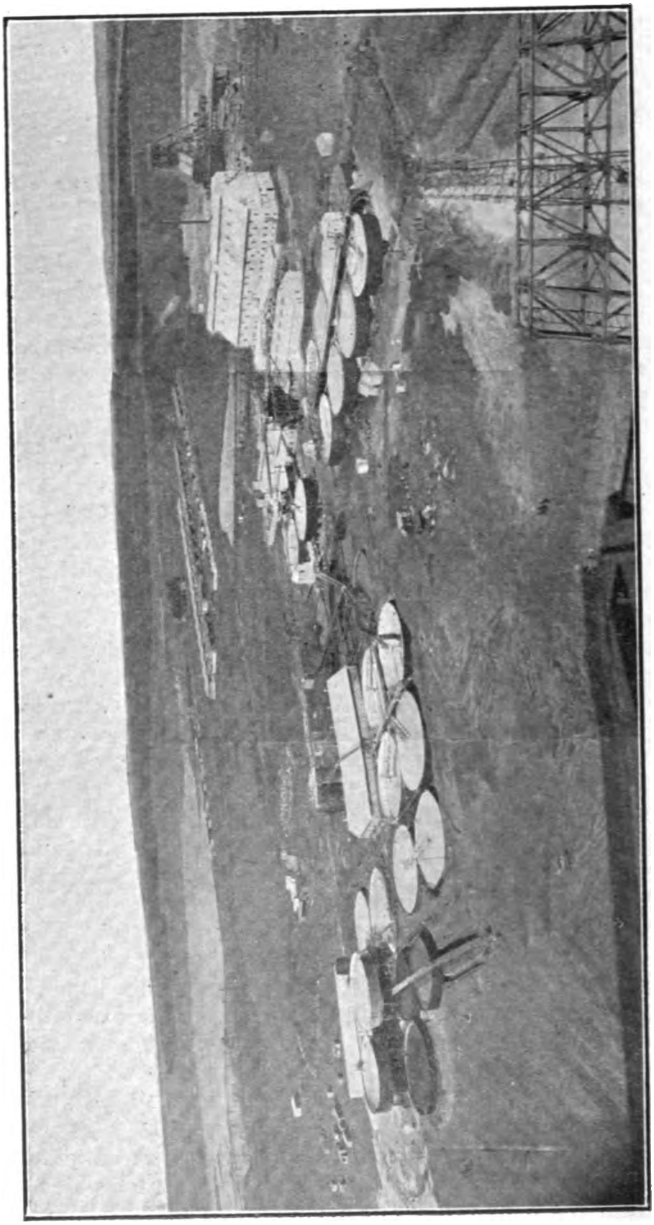
(1) Heavy stamps (2000-lb.), supplemented by eliminating screens at the back of the mortar-boxes; (2) multiplicity of service with the same centrifugal pumping units; (3) amalgamation following the tube-mill grinding only; (4) amplified and thorough classified, with simplified arrangements; (5) improved and simplified tube-mill feed; (6) sand treatment in accordance with the practice of separate collection, belt transfer, and the Blaisdell system of distribution in the treatment vats; (7) simple, direct, and efficient means of separating and conveying material; (8) aerial system of

tailing disposal; (9) power consumption separately metered and kept relatively low throughout.

A surface tram runs from No. 1 shaft to No. 2, and delivery of ore, as hoisted, is all made at a point immediately adjacent to the latter. Here the fine ore from No. 2 is discharged into a hopper and thence to a conveyor to the mill, which is also fed from the fine-ore bin at No 1. The coarse ore from both shafts is dumped into a large holding bin. From this bin six parallel belts, five of which are in use, were provided for in the plans, to carry the material to the sorting and crushing plant, and each belt delivers its burden to a revolving trommel, whence the undersize falls to a feeder-belt of the mill-conveyor, and the reject passes to belts discharging to the crushers. From the latter, which are equipped with a dust-removal system, the product is taken to the same mill-conveyor carrying the fine ore from the shafts and the trommels. In the sorting shed are belts parallel with and slightly lower than the conveyors, upon which the waste rock is thrown; and these discharge at the end to a large belt-conveyor, running at right angles, which carries the waste to a bin; whence it can be hauled along the dump.

The main fine-ore conveyor, rising to the top of the mill, discharges on a transverse belt, extending the width of the 600-ton bin, which is provided with a tripper for evenly distributing the ore. From the bin the ore is delivered by suspended automatic feeders to 160 stamps, with heads 42 by 9½ in. and stems 14½ ft. by 4½ in. The mortar-boxes are of the open-front type. The stamps weigh 2004 lb. each, of which 1076 lb. is in the head and shoe, and 928 lb. in the stem and tappets, and they are driven in groups of ten each. The stamps are arranged in two rows, with room at each end for an additional battery, the ultimate capacity of the mill to be 200 stamps.

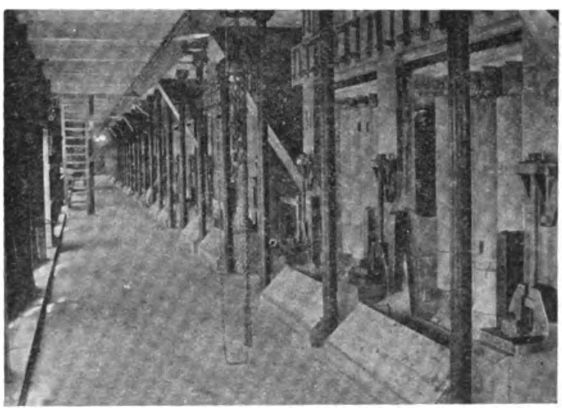
The ore is crushed through a coarse eliminating-screen, at the back of the mortar-box, whence it is forced by two sand-gravel pumps (Robeson-Davidson) through pitch-lined launders to Caldecott diaphragm-cone classifiers or pulp thickeners. These deliver a product which contains only about 27% moisture. The spigot flow from the classifiers drops into steel hoppers, from each of which it passes to a Schmitt spiral tube-mill. This is a feeder of a new type, originating on the Rand, which has already made a good record there. It is on the same general principle as the Pryce feeder, well known in South African practice except that the pebbles for grinding in the mill may be fed in by belt, or in any continuous manner. It also permits lowering the classifier cone, and this enables the tube-mill to be erected with less allowance for head. Furthermore, the 'bayonet' joint of the older type is replaced by a plain flange. The feeder is lined with silex blocks, except in the outer ring or receiving drum, where a removable steel plate is used to prevent the sand from packing and retarding delivery of material to the spiral casing. The pebbles are selected in



PANORAMIC VIEW, BRAKPAN MINES PLANT

the sorting shed from pieces of a hard flinty substance forming part of the banket ore and are delivered in cars over a return track.

The plans of the mill, with 200 stamps erected, provide for 10 tube-mills, or one to every two batteries, but the initial installation consists of 6. These are 5½ by 22-ft. Gates wet-grinding mills, 3 right-hand and 3 left-hand, made of heavy tank steel, with 6-in. silex lining. Manganese steel is used for the trunnion liners and ring liners of the feed end, and also for the discharge screen, these parts being the ones most subject to wear. In grinding, enough water is added to the contents to bring the moisture in the pulp up to 35%. Each machine is operated by spur gearing. This is placed at the outlet end to avoid the crowding together of the classifying, feeding, and driving arrangements at the head of the tube-mill. Each tube-mill discharges, through suitable launders, over five 5 by 12-ft, copper-plate tables, sloping 2.16 in. per foot. In the end



MORTAR BOXES

of each table, and in the launder at the end of each set of 5 tables, is a cast-iron amalgam trap. The pulp from these plates flows to a sump, whence it is raised by Robeson-Davidson rotary pumps to return cone-classifiers, the overflow of which goes directly to the cyanide plant, while the spigot product is returned to two of the tube-mills for regrinding. The grinding machinery and accessories were erected under the supervision of Herbert Ainsworth, the South African representative of the Allis-Chalmers Co.

In the plans, allowance was made for four 60-ft. sand-collectors, three of which are now in service; and, by the interposition of suitable Caldecott cones, a relatively barren sand is delivered to them, the slime overflow being taken away through launders. Subsequently the remaining content of the sand is extracted as far as possible in 6 (of an eventual 10) treatment-vats, and the treated sand is discharged by hand into buckets, underneath the vats, of 22-cu. ft. capacity, holding about one ton, and automatically dumped

by the Bleichert system of aerial sand haulage from the top of a cantilever (see illustrations), 210 ft. high, rising at an angle of 32°, driven by a 75-hp. motor. Adolf Bleichert & Co., of Leipzig and London, installed this system under a guarantee to dump sand at a cost not to exceed 3c. per ton. The buckets run on a monorail, are automatically attached to an endless rope, and can be dumped at any point on the circuit. The cantilever can be extended when the dump reaches the top.

The overflow from the sand collectors, and that from the cone-classifiers first above referred to, is re-treated in a third set of cones, the sand from which is returned to the collector cones. The slime is distributed to four settling-tanks and thence pumped to six Pachuca tanks, each 15 by 45 ft. After 4 hours' agitation, the direction of the pump suction is reversed by by-passing and the valves in the pipe-line set so that the pulp flows to a stock-pulp tank.

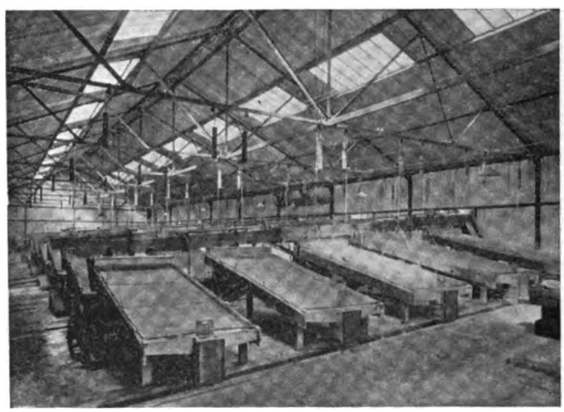


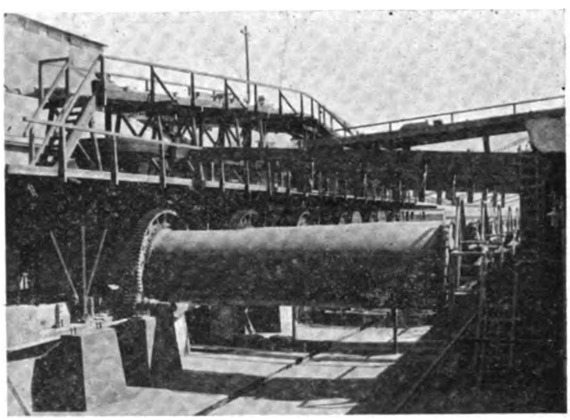
PLATE HOUSE

From the latter it is taken for treatment in a vacuum-filter plant of the Butters type. At this point there are placed the stock-solution tank, stock and surplus-pulp tanks, solution-storage tank, and others required in the system. There are two sets of Butters filter-boxes, each containing 6 sections of 28 leaves each, or a total of 336 leaves. Each leaf is 5 by 10 ft. A vacuum-pump draws the gold solution through the filter-frames.

Two sand-filters, each used alternately, are provided for clarfying the solution from the Butters filter, which is then delivered to the gold-solution storage and afterward pumped to filter-presses in the refinery. As the solution is drawn off zinc dust emulsion is fed into it automatically and continuously. The filter-presses, of which there are three, each containing triangular frames, are of the type invented by C. W. Merrill, who first used the system when metallurgist for the Homestake mines, Lead, South Dakota. From these the gold-bearing zinc slime is conveyed to acid-vats and sub-

jected to the customary treatment. Adjacent to the Merrill presses is the plant for recovering the gold from the amalgam and precipitate, and the apparatus for refining. The equipment includes, in part, a reverbatory-furnace capable of taking twenty No. 100 pots, and a calciner to take eight 3 by 2 by ½-ft. trays. There are three barrels for black sand treatment, whence the product passes to a batea amalgamating-plate and 2 by 2 by ½-ft. settling-bins. The amalgam from the batea is pressed in a couple of 4½-in. presses, and sent finally to a set of standard Fraser & Chalmers retorts. It is a noteworthy feature that, in the construction and equipment of the entire plant for ore reduction, the estimates originally made by the engineer were closely approximated, an excess in any one direction having been balanced in another.

In June of this year, when over 2,000,000 tons of 'payable' ore and half as much lower-grade ore had been blocked out, with the



TUBE-MILLS

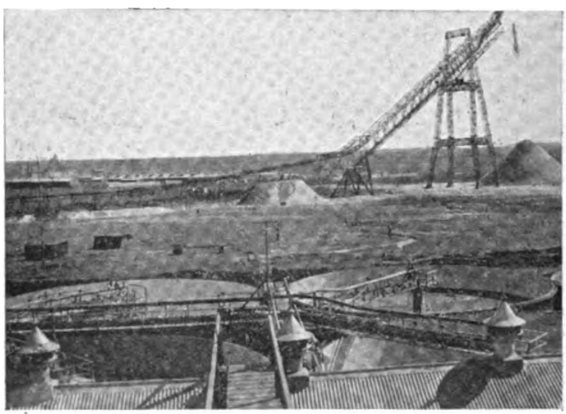
levels 500 ft. apart and stoping well in progress on all except the 9th and 12th levels, the stamp-mill made a preliminary run of 18,133 tons, yielding 4310 oz. of fine gold, valued at £18,087, the low yield being due to the absorption by amalgamating-plates and in cyaniding usual to the starting of a new plant. The working costs, including all overhead burden and 1s. 6d. per ton milled for development, the same as for all figures given below, amounted to £20,936. An operating loss of £2849, therefore, resulted.

In the following month, July, the figures were as follows: 100 stamps and 3 tube-mills running; 27,906 tons of ore hoisted and 4626 tons taken from the dump, representing stoping and development; with waste sorted, 25,400 tons milled; fine gold recovered, 9594 oz., value £40,352, or 31s. 9d. per ton milled; working costs, 21s. 10d. per ton milled; profit, 9s. 11d. per ton milled. Owing to certain favoring conditions not likely to be duplicated in subsequent operations, this recovery was conservatively rated by the

management as above normal. But in August the record again improved. The ore milled, with 100 stamps and as many of the tube-mills as required, running 25 days, was 35,051 tons; ore hoisted, 27,946 tons; ore from dump, 9401 tons; waste sorted; fine gold recovered, 12,149 oz., value £51,102, or 29s. 2d. per ton milled; costs, 17s. 5d. per ton milled; operating profit, 11s. 9d. per ton milled.

In September, with 100 stamps and 6 tube-mills in operation 28 days, 40,753 tons of ore was milled as follows: ore hoisted, 38,173 tons; ore from dump, 9650; waste sorted; fine gold recovered, 13,139 oz., value £55,071, or 27s. per ton milled; operating costs, 16s. 10d. per ton milled; operating profit, 10s. 2d. per ton milled.

During October, with 100 stamps and 6 tube-mills in operation for 30 days, 42,089 tons of ore was milled, 43,176 tons having been hoisted, 6889 tons taken from the dump, and the waste sorted out.



TAILING STACKER

The fine gold recovered was 13,125 oz., valued at £55,070, or 26s. 2d. per ton milled. The operating costs were 16s. 7d. per ton, yielding a profit of 9s. 7d. per ton milled.

The management of the Brakpan Mines, Ltd., is to be commended for the fact that, besides blocking out large ore reserves in advance of milling, it has apparently not attempted to make an impression of economy by niggardliness in expenditure. On the contrary, the construction and equipment all bears evidence of having been planned with a view to permanent investment, so far as any mining enterprise can be considered permanent. Building materials, for example, were mainly restricted to iron, steel, concrete, and brick. Furthermore, ample provision is made for repairs and maintenance, and for the welfare of the employees, including a native compound and hospital, and the company bears its share in all work for the good of the district as a whole. It is also a member of the Consolidated Mines Selection Company's Recruiting Agency,

and has been successful in obtaining a good grade of Kaffir labor. Among contributions made with other companies is one for the maintenance of the Miners' Phthisis Sanatorium.

For the excellent results already obtained, credit is shared with F. L. Bosqui and W. L. Honnold, by their assistant, C. E. Knecht, J. F. Cook, consulting mechanical engineer, A. J. R. Atkins, mill superintendent, and especially the general manager, C. B. Brodigan, as well as others to whom supervision of the various details has been entrusted. When operating to its maximum capacity, as sufficient ore is mined, the plant will exceed the estimate of 60,000 tons per month originally made for it, particularly in so far as the stamps and tube-mills are concerned; and it is reasonable to believe, from the experience already had, that the remainder of the equipment will do its proportionate part.

KUK SAN DONG CYANIDE PLANT

By A. E. Drucker

(January 13, 1912)

*The method of treatment consists of washing, classifying, and re-concentrating the material as it comes from the dump. The tailing from the concentrator plant is rejected, while the concentrate obtained, containing the bulk of value, is classified, re-ground (all-slimed), and the gold extracted by cyanide agitation and filter-pressing. The plant consists of a concentrator and cyanide annex. The concentrator contains one water supply tank (12 by 12 by 10 ft.), one storage dump bin (22 by 20 by 12 ft.), one 9-in. screw conveyor, one 5 by 3-ft. revolving screen washer and distributor, one 36-in. hydraulic cone classifier, four Callow 6-ft. cone thickeners, and eight No. 5 Wilfley tables. The cyanide annex contains one steel (15 by 12 ft.) stock cyanide solution tank, one concentrate storage bin (22 by 10 by 12 ft.), two 6-in. bucket elevators, one 6-in. screw conveyor, one 4 by 16-ft. tube-mill, two 3-in. Morris centrifugal sand-pumps, two hydraulic 36-in. cone classifiers, two 6-ft. Callow cone thickners, four Pachuca steel agitators (8 by 30 ft.), two steel sand-clarifying tanks (10 by 6 ft.), four zinc-precipitation boxes (36 by 30 in. by 12 ft.), two steel cyanide-solution sumps (15 by 12 ft.), two Morris 2-in. centrifugal solution-pumps, one Monteju pressure tank, one Dehne (24-frame) filter-press, one vacuum-pump, and gold precipitate filter-box. The power-plant consists of one 90-hp. steam-engine, one air-compressor (300 cu. ft. per min.), and one Fraser & Chalmers (60 in.) steam boiler. The foregoing is a full description of the plant. It occupies a space 30 by 140 ft., with an engine and boiler room annex. The tube-mill, pump, engine, and Wilfleys are set on concrete foundations. It was necessary to put in four stone retaining walls.

Up to August 1, 1911, the plant had a very irregular run, due chiefly to trouble with the sand-pumps, Pachuca agitators, steam-

*From report of the Oriental Con. M. Company.

engine, and inexperienced native help. The Pachuca agitators were connected in series and fitted with baffle cylinders so as to use the continuous agitation and decantation method which has proved an advantage in the modern plants of Mexico. This arrangement has proved a failure in our case, neither were we able to obtain a satisfactory circulation of the pulp, nor clear solutions to the clarifier tanks and zinc-boxes. This has been the most serious trouble and has prevented the getting through of a satisfactory amount of concentrate, also interfering largely with precipitation in the zinc-boxes. We have, however, at the present time overcome these defects by a different arrangment.

It must be remembered that in no other place has the attempt been made to re-treat an old oxidized concentrate cyanide tailing dump by the cyanide process, and in no case have we been able to profit by the experience of others on a similar proposition. Therefore we have had to work out a special treatment of our own. The plant is now treating 60 tons per day and will be brought up to 80 tons capacity per day, that which it was originally intended to treat. During the past we have been running through dump that only assays $2.56 per ton, so as to prevent losses until things were made right. Our new plant at Taracol will reap the benefit of our experience at Kuk San Dong. The assays and bullion returns so far show that we are able to obtain the following results:

Dump heads, $2.56 per ton (average heads for total dump, about $7).

Gold extraction by concentration, 79% (value contained in concentrate).

Gold extraction by cyanidation, 73% (value extracted from concentrate).

Total bullion extracted from $2.56 heads, 58 per cent.

OPERATING COSTS AT THE PITTSBURG-SILVER PEAK MILL

(February 10, 1912)

A complete description of the mill of the Pittsburg-Silver Peak Gold Mining Co., at Blair, Nevada, was prepared by Henry Hanson, at that time mill superintendent, and published in the *Mining and Scientific Press* soon after the plant was completed.* Detailed statements of mining costs, which for desert conditions are notably low, were furnished within the past year† by Edmund Juessen, till recently general manager for the company. Below are given corresponding details covering operating costs in the mill for the year 1911. These are printed through the courtesy of S. J. Kidder, general superintendent, who has immediate charge of the mill.

Mining and Scientific Press, May 8, 1909, republished in 'More Recent Cyanide Practice,' pp. 263-273, 1910.

†*Mining and Scientific Press*, July 8, 1911.

The general plan of ore treatment is essentially the same as when it was described by Mr. Hanson, though 20 stamps have been added, making the total now 120, and some changes have been made in the handling of solutions and other minor matters. The process involves crushing to 35 mesh with 1050-lb. stamps, 96 drops per minute, with a fall of 6½ to 7½ in. Outside amalgamation is used with a plate area of 12.8 sq. ft. per stamp, and a grade of 1¼ in. per foot. The plate tailing goes to a distributing sump and thence to sizing cones, 4 ft. 3 in. diam., fitted with Merrill hydraulic sizers. Before classification the pulp carries 5 or 6 parts of water to 1 of solid. After classification the sand contains 3 or 4 parts of water and the slime 14 to 16. The leaching-vats are five in number, 36 by 11½ ft. The whole cycle of treatment from charging to charging occupies a little over seven days. The combined overflow from the settling and sizing cones flows to five dewatering tanks. The thickened slime is drawn continuously from a bottom outlet and flows to a sludge storage tank where cyanide and lime are added to make an 0.025% cyanide solution. The pulp is agitated constantly by air and drawn continuously to Merrill filter-presses. Zinc dust is used for precipitation. The refining process used at the plant is practically identical with that used at the Homestake mine in South Dakota.

All men employed in mill and cyanide plants work 8 hours. The regular mill-crew consists of 3 men on a shift. In addition to this number are 2 plate-men and 2 repair-men on the day shift, making in all 13 men for the 24-hr day. In the cyanide department 4 men are employed on each shift, with no extras on the day shift, making the total cyanide crew 12 men for the three shifts. In the refinery 2 men are employed, on the day shift only. Their work consists of cleaning the precipitation-press and carrying on all work connected with the refining of mill and cyanide products. In the assay office, where the assaying for the mine and plant, as well as some custom work, is done, 3 men are employed. The average cost per ton for mining, milling, and all expenses in December, 1910, was $2.60.

The average cost per ton for each step in the milling process in 1911 was as follows:

Stamping	$0.298
Amalgamating	0.047
Neutralizing and settling	0.074
Leaching and sluicing	0.145
Filter-pressing	0.104
Precipitating	0.036
Refining	0.048
Assaying	0.033
Water service	0.070
Heating	0.007
Superintendent and foremen	0.053
Total direct operating	$0.915
Pro-general	0.079
Suspense account	0.046
Total operating	$1.040

The use of a 'pro-general' and 'suspense' account, is explained in Mr. Juessen's article to which reference has already been made.

PURISIMA GRANDE MILL, PACHUCA

By E. Girault

(March 23, 1912)

*This mill, designed and built under my supervision, is at the place once occupied by the oldest *hacienda de patio* of Pachuca, where it is said that Bartholomé de Medina invented in 1557 the amalgamation process. Some interesting features of the new mill are that it is built on practically level ground and that Chilean mills are used as primary grinding machines; four of these were taken from the old mill. These four Chilean mills are the only old machinery used in the new construction. About 100 tons of ore is put through the mill per 24 hours; the general plan of operation consists of fine grinding, concentrating, agitation in Pachuca tanks, filtration with Moore filters, and precipitation on zinc-shavings.

Ore Treated.—The ores are from the Guadalupe and Fresnillo mines; ore coming from the lower levels contains an average of 75% silica, 10% lime, and 7% iron. The ore from the upper levels carries oxides of manganese and iron; and although it proved refractory in the old patio process, it causes no trouble in cyaniding.

Grinding.—There are one 15 by 9-in. Blake crusher, and five Krupp Chilean mills with runners of 1.5-metre diameter, weighing with new tires 4080 kg. each. The runner shafts can move up and down independently, by which arrangement the wear of the tires and dies is uniform. The five Chilean mills are driven by a 100-hp. motor and have a speed of 18 r.p.m.; the power consumption of each mill varies from 15 to 20 hp., depending on the hardness of the ore and the proportion of fine and coarse.

The consistence of the pulp in the mills is 1 of solid to 6 or 7 parts of solution of 0.2% cyanide. Nine kilograms of lime is added per ton of ore. The mills are provided with No. 4 screens, the discharge is 10 cm. above the bottom, and the depth of the pulp is 20 centimetres.

The average capacity per mill per day is 24 tons when the feed consists of 78% fine and 22% coarse, calling fine that which does not exceed 5 cm. Recent sizing tests of the discharged pulp showed:

	Per cent.
On 100-mesh	38.2
On 150-mesh	9.5
On 200-mesh	5.1
Through 200-mesh	47.2

*Abstract from the *Informes y Memorias* del Instituto Mexicano de Minas y Metalurgia by G. Witteveen.

The tires and dies are of manganese steel. The tires, when new, weigh 1660 kg.; they last six months, and at the time of renewal weigh 400 kg. The dies weigh 1500 kg.; they last three months and when worn down to 300 kg. are renewed. The consumption of iron is 830 gm. per ton crushed, which is three or four times as much as in stamp-mills. Taking into account the total iron and steel consumed, it amounts to 1.054 kg. per ton of ore crushed.

Concentration.—The discharge from each mill goes over one Wilfley table. The extraction on these tables is about 18% of the content in ½% the weight of the ore. At the same time part of the classification of the pulp takes place on the tables, some slime being sent directly to the thickeners.

Classifiers.—There are two Dorr classifiers.

Tube-Mills.—Of the two No. 5 Krupp tube-mills, 1200 by 6000 mm., only one is in operation at this time. They are provided with El Oro linings, cast in Pachuca with 25% manganese steel from old tires and dies. These linings weigh 6 to 7 tons and last more than a year; the price is $0.12 per kg. Instead of flint, hard 10-cm. pieces of rock from the mine, selected from the dump, are used. The daily consumption is about 1500 kg. The Tube-mill discharge is pumped up about 14 ft. to the Dorr classifiers by means of a 10 by 54-in. Frenier pump.

Centrifugal Slime Pumps.—The slime is pumped 20.5 metres to the thickener by means of No. 2 Traylor centrifugal sand-pumps. Each pump has a capacity of over 100 tons of slime per day, the slime having a consistence of 1 of solid to 7.7 of solution. Each pump is driven by a 20-hp. motor and consumes 16.3 hp. They are fed under pressure through a conical receiver of 6 ft. diameter. The pumps have been in operation almost one year without causing the slightest trouble. The Traylor pumps make 1700 r.p.m. and can lift slime of 1 to 4 solution to a height of 20.5 m. The mechanical efficiency by actual measurement is 18.4%. The linings are cast in Pachuca; they are set with cement and last two weeks. The shafts last three months, the packing three weeks. With the exception of the high power consumption the pumps are satisfactory.

Thickener.—One 30 by 12-ft. Dorr thickener handles easily more than 100 tons of the thickening pulp from 1:1.7 to 1:1.44. A sizing test of the discharged pulp showed:

	Per cent.
On 100-mesh	12.27
On 150-mesh	8.50
On 200-mesh	3.95
Through 200-mesh	75.28

Pachuca Tanks.—There are four 15 by 45-ft. Pachuca tanks, which are charged with 94 tons of slime and 136 of solution, of about 0.257% KCN. It takes a little less than 24 hr. to fill a tank. During the filling a moderate agitation is maintained. After the

tank is full, agitation by means of the air-lift is kept up for 36 hr., using air at 25 lb. pressure. The tank is then left to settle for about four hours. A series of comparative tests showed not only that lead acetate is not necessary, but that the extraction was not as good and the cyanide consumption higher when using acetate. With acetate the cyanide consumption was 676 gm. per ton and the extraction 73.40%, against 524 gm. per ton and 74.7% without it. These results coincide with those obtained in other small mills in Pachuca, but not with San Rafael practice, where a constant use of acetate is necessary to get a good extraction. The reason for this difference seems to be that in the small mills, all the solution is passed through the zinc-boxes, which is not possible in San Rafael, owing to the large quantity of solution in circulation.

Compressor.—The 300-cu. ft. compressor for agitation, filtration, and other purposes, has in some cases proved to be sufficient. The pressure of the air is 25 lb. Two tanks in active agitation consume 26.3 hp.; two tanks and the filter, 31 hp., which is also sufficient to keep the four tanks in moderate agitation. In the plans, the air consumption per tank was estimated at 100 cu. ft. per minute with a power consumption of 10 horse-power.

Filter Plant.—A Moore filter with two sets of twenty 10 by 6-ft. leaves is used. The contents of a Pachuca tank containing 95 tons of dry slime is filtered as follows: New filter-leaves, 20 cycles in 12 hr. with 1 to 1¼-in. cake; half-worn leaves, 23 cycles in 14 hr. with ¾ to ⅞-in. cake; old leaves, 30 cycles in 20 hr. with ½-in. cake. Taking the average of 62 tanks, the unwashed cake contained 50 gm. of silver per ton and the washed cake 49 gm. The water of the discharged cake has 2.5 gm. silver per ton.

Precipitation is effected in five zinc-boxes of five 3 by 3 by 2-ft. 9-in. compartments. No. 9 zinc is cut into shavings 0.15 mm. thick. The zinc consumption is 1.450 kg. of precipitated silver.

Melting.—There are four coke furnaces, 2 ft. 2 in. inside diameter and 4 ft. high, with 2-ft. sheet-iron chimneys 80 ft. high. Either No. 300 Dixon graphite crucibles or Morgan crucibles with clay lining are used. The precipitate, air dried in the Shriver press, contains 8 to 14% moisture and is melted with the following fluxes:

	Kg.
Moist precipitate	100
Ground glass	10
Ground borax	6
Carbonate of soda	8

The first melt lasts about 6 hr., the next one 4 hr. The total cost of the mill, including what has been adapted from the old mill, has been about ₱1500 per ton per day capacity. The total power consumption is 2.07 hp. per ton of ore. From the results obtained in this mill, it seems that with a small mill the installation cost of the combination slow-speed Chilean mills and tube-mills is less, and that the operating cost and the extraction are about the same as with stamp-mills.

ASSOCIATED MILL, MANHATTAN

By J. C. KENNEDY

(July 27, 1912)

The Associated mill at Manhattan, Nevada, is owned by the Manhattan Associated Milling Co., of which John G. Kirchen is president. The mill was designed and erected under the supervision of Charles Kirchen, who is general manager for the company; R. W. Marston, formerly of the Tonopah Extension mill, is superintendent. Construction was begun late in November, 1911. The usual, or greater than usual, number of delays occurred in getting in machinery and construction material during the winter season, but the mill started running April 1, 1912, the day set. Less than the usual amount of trouble occurred on starting the mill, which speaks well for the experience and skill of L. A. Blackwell, the millwright and foreman in charge of construction and installation of machinery.

The principal stockholders of the owning company are interested in the nearby White Cap, Steffner Consolidated, and Mushett leases. The mill is so situated with reference to the leases mentioned, as well as the Kendall & Douglas, on the Manhattan Consolidated, the Swanson lease on the Earl, and the Bath Bros. leases on Litigation Merger and Earl, and the workings of the Manhattan Amalgamated Mining Co., that the output of all these leases and properties can easily be sent directly to the main upper sampling floor of the mill. So far, ore from the Steffner and the Mushett leases has been trammed to the mill, while that from the White Cap has been brought in large ore-wagons, mainly on a down-hill grade. Ore-bins delivering to cars on the tram track are erected outside of mill for different lots of ore. The mill is designed for general custom work in the district in addition to milling these ores, and for that purpose a fairly complete automatic sampling system has been installed. Ores will be bought and paid for in small or large lots after sampling and assay. This is a new departure in custom milling in the district which is pleacing and beneficial to sellers of small lots of ore, and contributes toward lower milling costs by avoiding delays for frequent clean-ups and consequent waste of time and labor. It is said that unusually low milling rates for this section are offered to owners of small lots of low-grade ore. So far the mill has had an abundant supply of ore from two or three of the leases, and this seems to be the outlook for the immediate future.

Fresh water for the mill is collected in a 20 by 16-ft. wooden tank, placed at such an elevation above the mill as to give a pressure of about 75 lb. at the filter floor for discharging the cake. This water comes mainly from workings of the Steffner and White Cap leases. Owing to the light winter rainfall, the water supply is rather short at present; but as the greatest depth yet attained at

either lease is only about 200 ft., no anxiety exists for the future. In a well in the gulch below the mill, water is collected from the drainage of the tailing ponds and pumped to the mill to reinforce supply. Other sources of supply are easily available.

The ores of the White Cap, Steffner Consolidated, and, to a less extent, that from the properties and leases farther west on the mineral zone, while apparently fully oxidized, are refractory. The White Cap ore only yields on the plate about $1 per ton from ore $50 or higher in value, and the Steffner ore not much more. Much of the gold is in a very fine state, coated in some manner which would indicate the necessity of fine grinding. A recently discovered ore-shoot in the Mushett lease, however, carries much coarse native gold in reticulated and crystalline forms. The White Cap ore contains a small amount of cinnabar and other sulphides. The Steffner ore contains sulphides of antimony and arsenic. These sulphides have so far not proved objectionable as cyanicides.

There are two steel storage tanks, 9 by 14, and a wooden tank, 12 by 16 ft. The solution is kept at a strength of about 1.5 lb. KCN per ton. The main upper or sample floor of the mill is provided with steel plates for coning and quartering down large samples. The southeast corner of this floor is partitioned off as a room for crushing, pulverizing, and otherwise preparing samples for assay. The assay laboratory itself is in the office building a short distance from the mill. The ore is first weighed and then dumped about 4 ft. to mouth of crusher, with no intervening grizzly. The crusher is of the Foster type. With the ores so far handled, which have a considerable percentage of friable and earthy matter, the capacity of the crusher is fully 50 tons in an 8-hour shift.

The crushed ore is raised by an elevator to an automatic sampler of the Vezin type, which cuts out a 1/20 sample, the reject falling driectly into a flat-bottomed ore-bin of 80 tons capacity. The sample passes to a set of rolls, thence to a second Vezin sampler, which takes out 1/10 for a final sample, which falls into a receptacle in a locked compartment at the end of battery bin. The reject falls directly on sampler floor, is transported in cars, dropped through the crusher when not in motion, elevated, and dropped into the battery-bin. Crusher, elevator, rolls, and sampler are driven by a 30-hp. motor. The ore is discharged from the bin through the usual rack and pinion gates to suspended Challenge feeders to the mortars of a 10-stamp Hendy mill, driven by a 30-hp. motor. The stamps weigh 1050 lb., with from 7 to 7½-in. drop at the rate of 100 drops per minute. The mortars are of the narrow quick-discharge type. Battery posts are large, and the concrete battery foundation and battery-frame are of the strongest and most substantial construction. All foundations in the mill are of concrete and of good size. Screens of 30 mesh or finer are used. The Tyler top-cap No. 50 is used for some ores, and the No. 273 for harder ores.

The pulp goes to a 30-in. by 15-ft. Akins classifier placed adjacent and parallel to the tube-mill and set on a slope of 2½ in. to the foot. The tube-mill is 4½ by 16 ft., with scoop feed, and is driven at 25 revolutions per minute by a 50-hp. motor. The discharge from the tube-mill goes to the classifier and the sand is returned to the tube-mill, thus forming a closed circuit; all the material traveling around the circuit until brought to such a fineness that it passes off in the overflow from the classifier. The tube-mill is provided with the Komata liner, which is rare and recent in Nevada, its previous use, so far as I know, being confined to the Goldfield Consolidated and Tonopah Extension mills. One of its features of merit is that it has a larger effective inside diameter for a given outside diameter, compared with silex blocks or other lining. Another is that the inside diameter varies less during its life, making the peripheral speed more constant, an important consideration. It is claimed that a less depth of loading with pebbles is required, the practice in New Zealand being to keep the top load of pebbles 5 to 7 in. below the axis of rotation. It is also quickly put in and removed, keeps its place, and wears well. It is being introduced in this country by F. C. Brown, and seems to be giving satisfaction.

All the lime required to neutralize the acidity of ore and keep up the protective alkalinity in the agitator is introduced into the classifier. This, so far, has been about 7 lb. per ton. Overflow slime from classifier is pumped by a 4 by 18-in. Byron Jackson centrifugal pump to a 10 by 20-ft. Dorr thickener, settler, and dewaterer. A 5-hp. motor drives the classifier, pump, and thickener. The overflow solution from the thickener flows to the gold-solution tank. Should it not be high enough in gold content it can be diverted either to the weak-solution tank or the barren-solution sump-tank under the zinc-boxes. The thickened pulp flows by gravity to one of two tanks 16 by 20 ft., fitted with the Trent agitator. At this point enough KCN is introduced to bring the strength of the solution up to about 2 lb. per ton. Some of the ores, notably the White Cap, settle rather badly, but an attempt is made to keep the specific gravity of the slime in agitators at about 1.2. Of this slime, 65 to 70% will pass 200 mesh. The slime solution in each tank is kept in circulation by a Campbell & Kelly centrifugal pump run by a 5-hp. motor. The usual period of agitation in each tank is 16 to 24 hours.

The Trent agitator is well known, but as it seems to be very effective and dispenses with the installation and operating costs of compressed air required in the Pachuca tank system, a brief mention will be made. The slime is drawn from the top of the tank and forced by the pump through a central pipe at the bottom. The central pipe has four pipes radiating from it, each fitted with jet-pipes directed toward the bottom of the tank. The reaction of the discharging slime through these nozzles causes the arms to revolve at sufficient speed to give good agitation. Air is admitted

into the suction of the pump and is forced through the slime charge, giving a well-disseminated aeration. Air is automatically trapped beneath an inverted hemispherical convering which surrounds the step-bearing, so that no solid or liquid matter can reach the bearing, making the friction not greater than if it were in the open air and not submerged.

From each agitating-tank, when agitation is completed, the slime is pumped to an 18 by 26-ft. pulp-storage tank, also fitted with a Trent agitator, by which agitation is continued to preserve homogeneity. From this tank the slime goes to a Butters filter-box, immediately below, which contains 45 leaves, 5 by 10 ft., with two hopper bottoms, and with the usual acid-treatment vat at one end. The slime enters the box through a large pipe, fitted with nozzles, which shoot the slime some distance out over the top of the frames. Adjacent to and on same level as the filter-box floor are two tanks, 10 by 16 ft., one containing weak solution and the other wash water. The cycle of operations is about the same as usual. One wash with weak solution and one with water are given when the vacuum is on, all except part of the water wash going to the gold-solution tank. In practice in this mill the cycle from commencing to load to the final discharge of the cake is longer than usual. With a ¾-in. cake the filter handles about 12 tons of dry slime at each cycle. The handles and levers of all the valves are conveniently arranged on the filter-floor, and all those for diverting slime, weak solution, and wash water in the desired direction are placed compactly in small space at one corner of the filter-box.

From the gold tank the gold solution goes to two steel zinc-boxes, each with 7 compartments, 3 ft. square and 30 in. deep, having hopper bottoms covered with screens to retain short zinc. The two head compartments in each box are filled with excelsior for clarifying purposes. These boxes are placed immediately above a large sump-tank into which the barren solution falls. Adjacent to this tank and on the lowest floor of the mill are a Goulds 4-in. wet-vacuum pump, 40 r.p.m., and a Meese & Gottfried Butters 6-in. centrifugal slime and solution pump.

The mill has an addition at the lowest level containing a boiler for steam-heating the mill and solutions when necessary. It also contains an Ostermoor pipe-cutting and threading machine, and other tools. One end of this addition is partitioned off and fitted with cement floor for clean-up, drying, and bullion room. All short motor belts in the mill are of the metal link-belt pattern. The frame of the mill building in anchored to concrete block foundations extending into bedrock; the sides and roof are covered with galvanized corrugated iron. The absence of vibrations about battery and in all parts of the mill is especially noticeable. The capacity of the mill is 50 to 60 tons per day; nearer the latter figure on present ores. The capacity of lower part of mill for almost every class of ore is considerably greater than that of the upper, unless agitating and filtering operations are unduly prolonged, thus giving considerable elasticity to the treatment of different kinds of ores.

TWO NEW TREATMENT PLANTS IN WESTERN AUSTRALIA

By M. W. VON BERNEWITZ

(August 3, 1912)

The Mountain Queen mine near Southern Cross, and the Yuanmi mine in the East Murchison, recently started the treatment of ores in rather interesting plants, and through the courtesy of J. A. Agnew, of Bewick, Moreing & Co., I am able to give a sketch of the methods of treatment.

The published yields from these two mines for April were as follows:

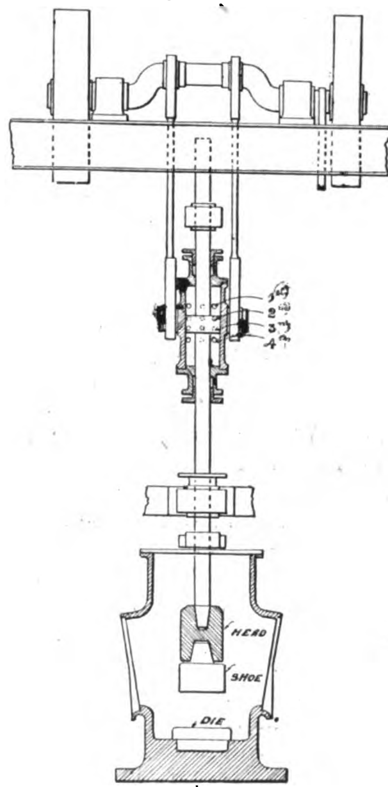
	Mountain Queen.	Yuanmi.
Tonnage	3,735	4,280
Yield	$17,200.00	$43,700.00
Profit	$5,600.00	$20,400.00
Mining cost	$1.20	
Milling cost	$1.06	4.32
General cost	$0.40	

The Mountain Queen has a set of double No. 1 Holman pneumatic stamps crushing the ore through 10 by 10-in. wire screens. These are the first pneumatic stamps at work in Western Australia, and I believe in Australia, though they are common in Cornwall, where they show a marked improvement in costs over the ordinary stamp. The pair under review crush from 135 to 160 tons of fairly hard ore per day, or 67 to 80 tons each. The speed is from 123 to 135 drops per minute, and, other things being equal, the pair is equal to a battery of fifteen 1250-lb. stamps. There has been no trouble with the patent gear, only the usual battery repairs are necessary, and the wear and tear is about the same as with ordinary stamps. Mr. Degenhardt, the firm's engineer, informed me that for cost of erection and general upkeep, the Holman stamp is very satisfactory.

The air-cushion is produced in the following manner: In the walls of the cylinder are four rows of holes (see figure), two above and two below the centre position of the cylinder. Rows two and four are plugged. As the cylinder travels up and down, it traps the air between the piston and the cylinder covers, and the air thus compressed acts as a cushion, and on the return stroke assists in the propulsion of the stem. As the shoes and die wear, the holes in rows two and four come into use, and rows one and three are plugged. By this patented device an equal distance is maintained between the shoe and die, for owing to the thinned 'air-cushion' being at the bottom of the cylinder, the piston, and consequently the shoe, is not raised as high as previously; a very considerable improvement on the old method. The following details are supplied by the manufacturers, and the weights given include steel framing:

Size	No. 1.	No. 3.
Diameter of cylinder (in.)	9½	6
Stroke (in.)	12	10
Power for stamp (hp.)	25	15
Single stamp, weight of battery (tons)	12¾	5¼
Double stamp, weight of battery (tons)	21	8¾

The pulp from the stamps in the Mountain Queen plant passes to two 5-ft. grinding pans fitted with Freeman classifiers, in which classification is by segregation due to centrifugal force. The flow from the pans then passes over two copper plates, 12 by 6 ft., and in



PATENT PNEUMATIC OR AIR-CUSHION ORE-STAMP

all, about 70% of the gold is recovered by amalgamation. The sand and slime flow to ponds, and will be treated later with the current mill product, by a new process, which is claimed to be of considerable interest. The plant described is driven by a 165-hp. Kynoch suction gas engine, which is powerful enough to drive the other plant being erected.

At the Yuanmi, the ore, after passing a jaw-breaker, is crushed by 20 Fraser & Chalmers 1250-lb. stamps, dropping 7½ in. 103 times per minute, at the rate of 8 tons per stamp daily through 10 by 10-in. screens. A larger screen, 10 by 12, is occasionally used;

depending on the character of the ore, which, from some portions of the mine, is very hard. Crushing is done in weak cyanide solution, this coming from the treatment plant. The mine-water is fairly fresh, containing the usual magnesia salts found in our northern waters. The mortars have a band shrunk around the screen opening to prevent cracking at this weak point. In this mill there is no amalgamation.

The battery pulp flows to one leg of a Forwood-Down 10 by 48-in. sand pump, and is elevated about 12 ft. to a cone classifier, 6 ft. diam., the underflow going to a 16½ by 4-ft. Krupp tube-mill, working at 29 r.p.m. The overflow from the classifier, as well as the discharge from the tube-mill, goes to the other leg of the pump. This is raised to another cone, and the underflow, which consists of coarse sand, returns to the first leg of the pumps, and through the tube-mill. The tube-mill is fitted with the corrugated liners so popular in Western Australia now, and a great quantity of the pebbles used consists of hard ore from the mine. The overflow from No. 2 cone-settler gravitates to two continuous mechanical thickeners, the overflow of clear water being returned to the battery-supply tanks. The thick slime flows to three ordinary agitators, agitated with cyanide, and finally treated in a gravity-type vacuum-filter, similar to that described by Degenhardt and Stevens in the *Monthly Journal* of the Chamber of Mines, for March of last year. The residue, containing 27% moisture, is mixed with mine-water, and flows to a pond. The first 15 to 20 minutes wash from the filter is passed through vacuum clarifiers, and then to the precipitating boxes.

The whole plant is driven by 200-hp. Crossley suction gas engine, with low cost. The simplicity of the Yuanmi mill is at once apparent, as there seems a tendency in some mills at present to be as complex as possible.

COLORADO MINE AND MILL, PHILIPPINE ISLANDS

By PAUL R. FANNING

(November 2, 1912)

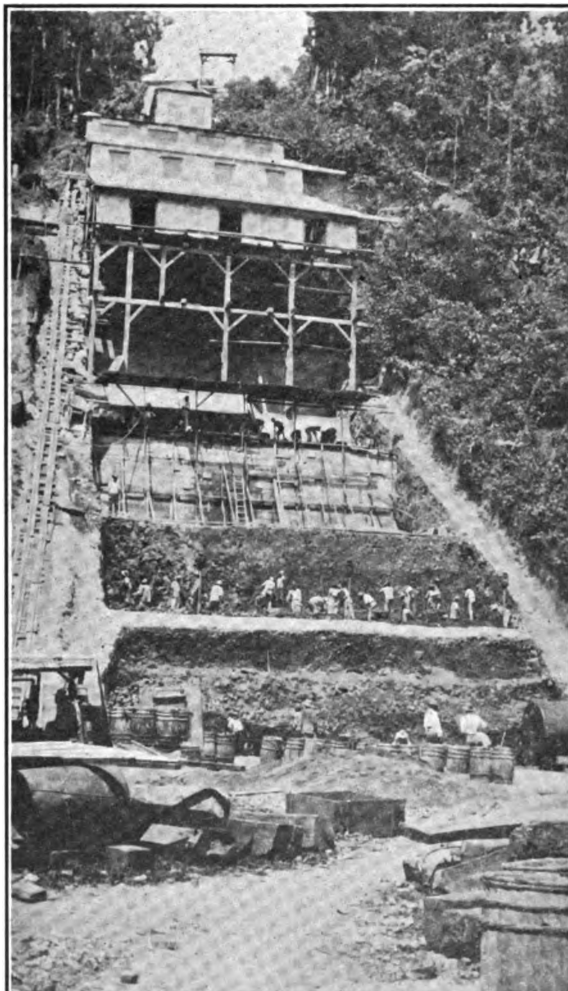
*During the first part of 1911 the Colorado Mining Co. continued development to prepare the mine for the 20-stamp cyanide mill which was nearing completion. During 1911 the mill operated irregularly for less than two months, but steady production did not begin until the last of December.

The Mine.—An accurate description of the mine and ore-bodies has already been given by Eddingfield. The vein averages 4 metres in width, strikes south 45° east, and dips 70° to the north-east. The vein matter consists of quartz, with some calcite and manganese. The ore is stated to average ₱20 per ton.

The inclined shaft now reaches a depth of 146 metres, which is the largest depth penetrated in the district, perhaps in the Philip-

*Extract from 'Mineral Resources of the Philippine Islands for 1911.' The peso is the Philippine peso, of a value of $0.50 gold.

pine Islands. As stated by the company, the average precious-metal content shows only a nominal decrease from the top to the bottom of the shaft, and the bottom ore is said to have a value of more than ₱16 per ton. It is to be noted that this ore is still above



COLORADO MILL DURING CONSTRUCTION

the level of the permanent ground-water, and hence in the oxidized zone. The indications are that the zone of oxidation should extend to a depth approximately equal to the level of the Guinobatan river; that is, 100 metres below the present bottom of the shaft. It is probable that good ore will be found to this depth, and it is

possible that the zone of oxidation may extend to an even greater depth. Whether or not a zone of enriched ore will be found at this depth is too problematic for present consideration, although the presence of manganese in the ore, as has been repeatedly pointed out, is highly suggestive of a zone of richer ore close to the level of the permanent ground-water.

The mine has been developed by means of two main adits, 53 metres apart vertically; the upper, or No. 1 adit, is 30 metres below the highest point in the outcrop. Two intermediate levels have been run and various cross-cuts and raises put in. During 1911 a drift was run next to the hanging wall parallel to and connecting with No. 1; the main, or No. 2, adit was continued 80 metres farther to the northwest; the upper intermediate level was run for a distance of 120 metres; the lower intermediate level was run for a distance of 180 metres; and the shaft was sunk 30 metres to a total depth of 146 metres. In all, some 800 metres of work was performed, besides preparing the ore-bodies for stoping.

In stoping the ore the square-set system of mining has been used, but owing to the high cost of timbering it is probable that some modification of the method will be adopted. The mine is surrounded by a forest of magnificent hardwood trees, but the cost of cutting, transportation, and dressing is very high. The tangle of underbrush and vines require an extra amount of work to fell the trees after cutting, and the wood is so hard that the natives saw it with difficulty. The sets are framed by hand and are so heavy as to require several men to handle each piece. The substitution of dressed pine shipped from the United States for native timbers is being considered as an economical measure.

The Mill.—The 20-stamp cyanide mill is of Traylor design and is of the all-sliming type, crushing in cyanide solution. The striking feature of the practice is that the gold solutions from the Dorr thickeners pass to the zinc-boxes without filtering in the Oliver filters. The ore is delivered from the second level of the mine to the storage bins, and thence to the mill by means of a 2-bucket aerial rope tramway 320 metres long.

Crushing.—The ore from the buckets passes to a 3 by 8-ft. grizzly; the oversize is crushed in a 10 by 20-in. Blake crusher and passes to the bin. The ore is fed into four 5-stamp batteries by Challenge feeders. Lime, which is prepared in a kiln near the mine from a local deposit of calcite ore, is added at the batteries. The 20 stamps, weighing 1250 lb. each, drop at the rate of 100 per minute; the height of drop is about 8 in., and a 4-in. discharge is used. The ore is crushed in solution of a strength varying from 0.12 to 0.20% potassium cyanide. A capacity of 4.5 tons per stamp per 24 hours is obtained when a 6-mesh screen is used. The pulp, which has a thickness of about 6 parts of solution to 1 of ore, passes to a Dorr classifier, where the sand is raked out and delivered to two 5 by 18-ft. tube-mills. These mills are of Traylor manufacture, with trunnion bearings and spiral feed. A speed of 28 r.p.m. is

used. The lining is of the El Oro type. At first, Danish pebbles were used entirely, but at present some of the harder quartz from the mine is added. The discharged pulp, which contains about 40% moisture, passes to a bucket elevator and is returned to the Dorr classifier.

Treatment of the Slime.—The slime, having a thickness of about 9 to 1, passes from the Dorr classifier to a 20 by 14-ft. Dorr thickener. The overflow from the thickener passes to a clarifying tank, thence to a gold-solution tank, and thence to three sets of 7-compartment zinc-boxes.

The pulp from the thickener passes to two 12 by 40-ft. Pachuca tanks, connected in series. The central columns of the Pachucas discharge at the surface and an air pressure of about 32 lb. is used. The solution has a strength of from 0.12 to 0.20% potassium cyanide, and the pulp has a thickness of about 1.5 to 1. After agitation in the Pachucas, the pulp passes by launder to a second Dorr thickener. Previous to entering the thickener the pulp is diluted with solution from the Oliver filters. The overflow from the thickener passes to a clarifying box, and thence to two sets of the 7-compartment zinc-boxes. When first operated the thickeners gave trouble owing to the great amount of slimy foam which overflowed with the gold solution. This difficulty was overcome by removing alternate bolts on the rim launder and drawing off the solution just below the surface free from foam. The foam is drawn off separately by a central launder and passes to the Traylor agitator.

The pulp from the thickener passes to a 20 by 8-ft. Traylor agitator, which prepares the pulp for two 11 by 6 by 8-ft. Oliver filters. From the filters the gold solution passes by means of wet and dry vacuum pumps to a 12 by 10-ft. storage tank. From this part of the solution is pumped to dilute the pulp entering the second Dorr thickener, and the remainder is pumped to the 28 by 14-ft. solution storage tank at the top of the mill. None of the solution from the filters passes directly to the zinc-boxes.

Solution of the Gold.—As noted, a feature of the practice is that the gold solution is taken from the Dorr thickeners for precipitation rather than from the Oliver filters. The reason for this lies in the fact that about 80% of the gold goes into solution before the pulp reaches the Pachucas. It would seem from this that the greater part of the gold is very fine, easily liberated, and readily attacked by the cyanide solution. The presence of manganese probably has been instrumental in producing this condition of the gold. Some coarse gold and gold bound up in the minerals are present, and the Pachucas give the added time necessary to their dissolution.

Precipitation and Clean-up.—A general clean-up is made every two weeks, and the bullion is melted once a month. The zinc fine is treated with sulphuric acid, and the gold slime is washed, steam-dried, and melted in a gasoline tilting crucible furnace. The bullion is shipped to Manila, where the International Banking Corporation advances a sum based on the company's own assay. The bank

ships the bullion to San Francisco, and later makes adjustments with the company.

Cost of Operations.—The cost of mining and milling at the Colorado mine has been placed at less than ₱8 per ton, and there is some reason to believe that this figure will be reduced under conditions of steady operation. The fact that the cost of mining is moderately low would lead to the belief that, considering his low salary, the Filipino is an efficient miner. It is to be remembered, however, that the vein is of such width as to permit stoping on an economical scale. Furthermore, most of the ore is sufficiently soft and fractured as to be readily mined without the use of dynamite. On the other hand, the cost of general supplies is high, although lower than at many mines in the United States not situated directly on a railroad. The anomalous condition of having a timber problem in the midst of a great forest has already been discussed, and this problem will become more acute as the trees are removed and it becomes necessary to bring timber from a distance.

Owing to the excessive cost of the mill, the original investment is large in comparison with the capacity, and this means a higher cost per ton than usual, owing to the factors of depreciation and interest. For this mill these two factors alone mean a cost of about ₱1 per ton, and it is possible that this amount has not been included in computing the total cost of operation.

It was my belief when first coming to the Philippines that the depreciation for this country would be especially high, and experience in many cases has justified this belief. However, there are instances where, under good construction, the depreciation has been no greater than in the United States. The destruction of buildings under the attacks of white ants or other insects does not always seem a serious consideration, as there are numerous buildings, properly constructed, which have lasted for many years; certainly many years longer than the probable life of any mine. It is also known that white ants do not attack timbers which vibrate under the shocks of machinery. The construction work on the Colorado mill is of such order that it appears unnecessary to allow an excessive depreciation charge.

MILLING PLANTS OF THE ORIENTAL CONSOLIDATED, UNSAN, KOREA

By A. E. DRUCKER

(November 23, 1912)

The gold-mining concession of the Oriental Consolidated Mining Co. covers an area of some 600 square miles on the headwaters of the Anju river, in northwestern Korea. Since the operations of the company are so extensive a number of stamp-mills and cyanide plants are required. The following account of the results secured through recent changes in construction at some of these plants will be of interest. Details of operating costs during the fiscal year

1910-1911, as given in the annual report, appeared in the Mining and Scientific Press of January 13, 1912.

Maibong Canvas Plant

This plant was designed to make a saving from the mill tailing. The method of treatment consists of a preliminary classification and reconcentration of the resulting fine and slime tailing on canvas-tables. Total cost of plant, which includes two cone classifiers, three Callow cone thickeners, and eight adjustable canvas tables (8 by 16 ft.) was $1871. The gross value of concentrate saved from the canvas table has been $9116 to July 1, 1912. The net profit equals $3290, after deducting cost of palnt, operating expenses, transportation, cyanide treatment of concentrate, and tailing losses.

Tabowie Canvas Plant

The treatment at this plant is almost identical with that at Maibong, with the exception that no Callow cones are used. The plant consists of eight cone classifiers and twenty canvas tables, which represents a total expenditure of $4354. The gross value of concentrate saved (243 tons) from the canvas since the completion of the plant (November 18, 1911) amounts to $5014 to July 1, 1912. This represents a total running time of five months, the plant being closed down during January and February on account of a shortage of water. The gross saving in concentrate for the month of July, 1912, amounts to $1200. Net profit to August 1, 1912, equals $97, after deducting the cost of plant and all other expenses. From now on I estimate an average monthly net profit of $800 to $900.

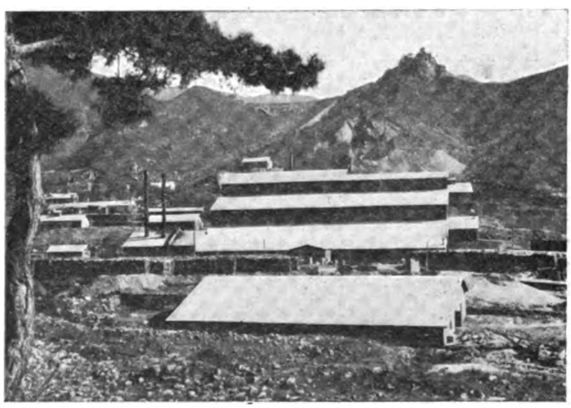
Taracol Canvas Plant

The treatment is similar to the other plants, with the exception that the coarse product from the classifiers is treated on Wilfley tables instead of belt vanners. The plant consists of four classifiers, eight Wilfley tables, and 28 canvas tables. The total cost of installation was $9461. The gross value saved in concentrate from the canvas tables, since the completion of the plant (March 11, 1912) amounts to $5898. Gross value saved during July equals $1696. The profit derived to August 1, after deducting all operating expenses, amounts to $4183. By January 1, 1913, this plant will be paid for, and from that time on I estimate an average monthly net profit of about $1150.

Kuk San Dong Dump Re-treatment Plant

The total time this plant operated during the past fiscal years was 212 days, and in that time 11,953 tons dump, 88 tons Maibong,

and 912 tons Kuk San Dong mill concentrate were treated. The method of treatment for the dump (sulphide tailing from old cyanide plant) consists of washing, classifying, and re-concentrating on Wilfley tables. The concentrate obtained, containing about 85% of the precious metals, is classified, re-ground (all-slimed), and the gold extracted by cyanide agitation and filter-pressing. Wilfley tailing from the concentrator plant is rejected. A description of this plant is omitted here, as it has already been described in my report for 1911. The concentrator plant (8 Wilfleys) treated an average of 56.3 tons per 24 hours of old dump material, or at the rate of 7 tons per machine. The average assay value of the Kuk San Dong re-treated dump material is $3.14 (this does not include re-treated Maibong tailing from old cyanide plant). The results obtained on this ($3.14) dump material are: gold extraction by re-concentrating, 85.7%; extraction from this concentrate by re-cya-



TABOWIE MILL

niding, 63.6%; net bullion extraction from dump, 53.9%. Considering the material treated and its low value, this result is up to expectations.

The material to be treated during the coming fiscal year averages $8.28, or $5.14 more per ton, and this increased value should net us a good profit. Our average total expense per ton for the dump material treated was $2.26, and this amount is less than the concentrate treatment expense ($2.81) at the Alaska Treadwell mines, where a similar plant treats 90 tons per day using cheap electric power. Upon a total tonnage of 12,953, including Kuk San Dong dump, Kuk San Dong mill, and Maibong mill concentrate, a net profit of $3881.14 was realized. My estimated net profit on the remaining Kuk San Dong old concentrate tailing dump, with practically all the low-grade material re-treated, according to the results already obtained, is as follows:

Cost of treating 29,056 tons at $2.26	$65,666.56
Original cost of the plant	39,336.38
Value remaining in tailing ($2.50 per ton)	72,640.00
Total	$177,642.94
Net recovery	63,014.42
Gross content (29,056 tons at $8.28)	$240,675.36

To the net recovery must be added $13,842 as credit to be received for the plant machinery upon completing the work, so that the ultimate net profit will be $76,856.42.

The foregoing report of the Kuk San Dong dump plant gives the results for the fiscal year ended June 30, 1912. The results for the first three months of the fiscal year 1913 have been most encouraging. They are as follows:

	Tons treated.	Gross content.	Bullion recovery.
July 1912	1,529	$8,219.23	$6,712.57*
Aug. 1912	2,060	7,753.47	7,103.80†
Sept. 1912	1,956	7,598.45	5,905.78
	5,545	$23,571.15	$19,722.15

JULY, AUGUST, SEPTEMBER 1912

Average value of material per ton treated	$4.25
Average actual net bullion extraction, per cent	76.47‡
Total time operating, days	73
Capacity per 24 hours, tons	76
Cost of treatment per ton	$1.75
Net profit obtained	$10,000

The following is an average analysis of the old cyanide tailing dump (now being re-treated) before it had become weathered and oxidized:

MAIBONG CONCENTRATE

	%
Gangue	62.96
Pyrite	17.60
Arsenical pyrite	17.00
Iron oxide	2.21
Galena	traces
	99.77

KUK SAN DONG CONCENTRATE

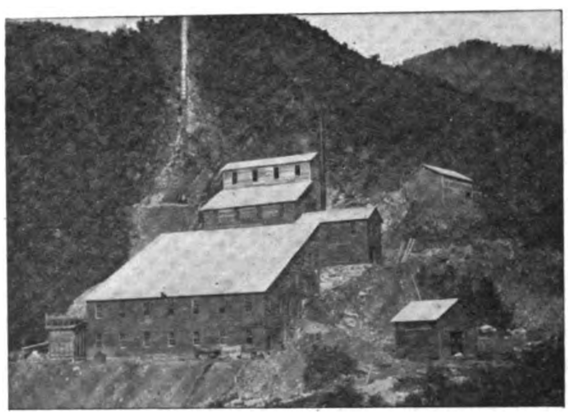
	%
Gangue	50.60
Arsenical pyrite	27.04
Marcasite	13.12
Galena	9.14
Sphalerite	traces
	99.90

(Gangue consists mainly of quartz.)

It is easy to imagine the difficulties likely to be encountered with such material after standing for eight to ten years before receiving a second cyanide treatment.

*Includes $1703.02=bullion from May and June slag.
†Includes $896.25=bullion from July slag.
‡Does not include $1703.02 July slag credit, which should have been credited to the previous fiscal year.

The increase in extraction over the previous fiscal year (1912) so far amounts to about 20%. The bullion now being obtained is derived from the re-treatment of tailing from the old cyanide leaching plant. I attribute the better results to some changes made in the treatment during the past three months. Another small air-compressor was installed, which enabled us to put into commission a fourth Pachuca agitator, allowing for a longer time of agitation and treatment. Finer grinding of the sulphide was resorted to (all passing a 200-mesh screen). The barren solution wash applied through the main filling channel of the Dehne press was responsible for the better results in filter-pressing. A preliminary treatment Pachuca agitator has recently been installed which agitates the concentrate from the Wilfley tables with lime-water and air. The concentrate is now alkaline before passing to the tube-mill to be ground in cyanide solution. This effects a material saving in cyanide and



CANDLESTICK MILL

prevents to a large extent the fouling of the solutions, and better extraction consequently results. The present foreman of the plant, G. C. Evans, is to be congratulated upon his good work.

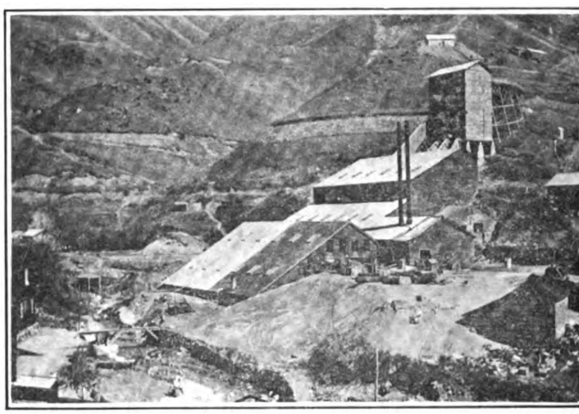
Taracol Cyanide Plant

The re-grinding plant was completed about the middle of May, 1912, but did not operate steadily until the latter part of the month. Even the following month of June was occupied mainly in 'breaking in' Koreans to the new work. Our mechanical troubles were due chiefly to inexperienced labor. Because of several days of lost time during June it was necessary to run several charges of concentrate into the old leaching plant to prevent an accumulation of concentrate at the Tabowie and Taracol mills. Consequently there were two plants running until June 15. The total operating expense

for both plants are included in one, and charged up against the new cyanide re-grinding plant. Also the total bullion clean-up was included as one and credited to the new re-grinding plant. The total concentrate for June treated in both plants was 2228 tons, which was valued at $55,993. The bullion extracted from this was $50,238. Percentage of extraction, 89.7; theoretical extraction, 87.4 per cent.

During July the total concentrate from both mills (160 stamps) was treated in the new re-grinding plant. The treatment consisted of classifying, all-sliming, continuous Pachuca-tank agitation, Merrill zinc dust in connection with zinc-box precipitation, and filter-pressing. One strong solution was used for the grinding and total treatment.

The plant consists of one 5 by 18-ft. tube-mill; two Morris 3-in. sand-pumps; two vertical triplex sand-pumps; four Pachuca agi-

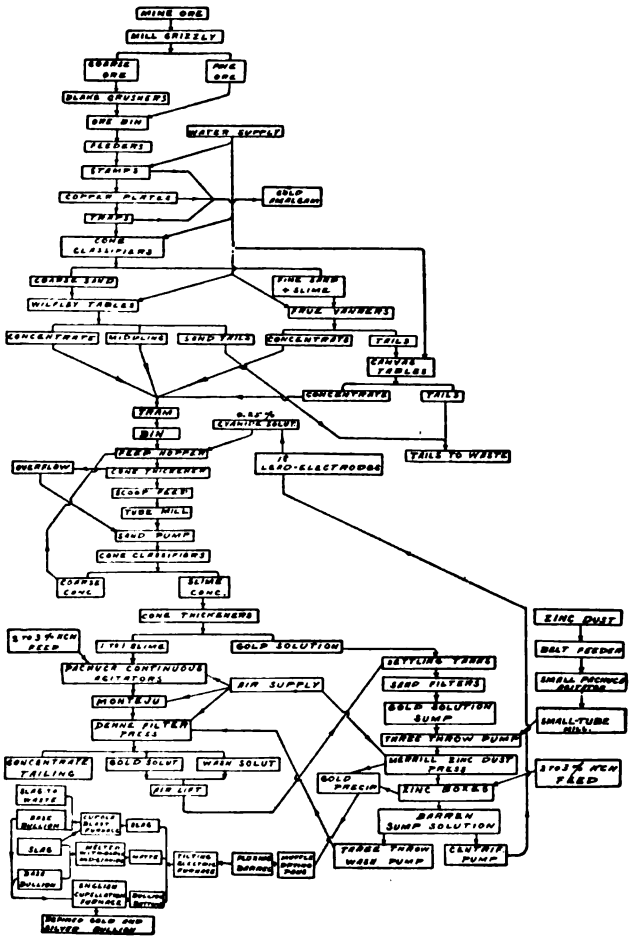


KUK SAN DONG REDUCTION PLANT

tators, 10 by 40 ft., connected for continuous agitation; four cone classifiers; six Callow cone thickeners; two settling tanks, 6 by 25 ft.; two sand-filters, 6 by 25 ft.; one Merrill zinc dust press; five zinc-boxes; one Monteju receiver; one Dehne filter-press, 50 frames; two Morris solution pumps; two vertical triplex pumps; one solution sump, 10 by 30 ft.; one gold-solution sump, 10 by 15 ft.; two stock-solution tanks; and a set of lead electrodes for the electrolysis of sump solution. The power-plant consists of one 250-hp. engine, three boilers, and two air-compressors, 500 cu. ft. and 107 cu. ft. air per minute.

The re-grinding plant results for the month of July are as follows: Total tons concentrate treated, 2070.3 of an assay value per ton of $29.20, or a total value of $60,446. The bullion obtained was

$53,314.87, corresponding to an actual extraction of 88.2% or a theoretical extraction of 88.9%. It must be remembered that no definite conclusion can be attained from the first month's run of a new plant such as this. Our original estimated extraction for this plant was 91%, and I am sure that after all parts have been adjusted

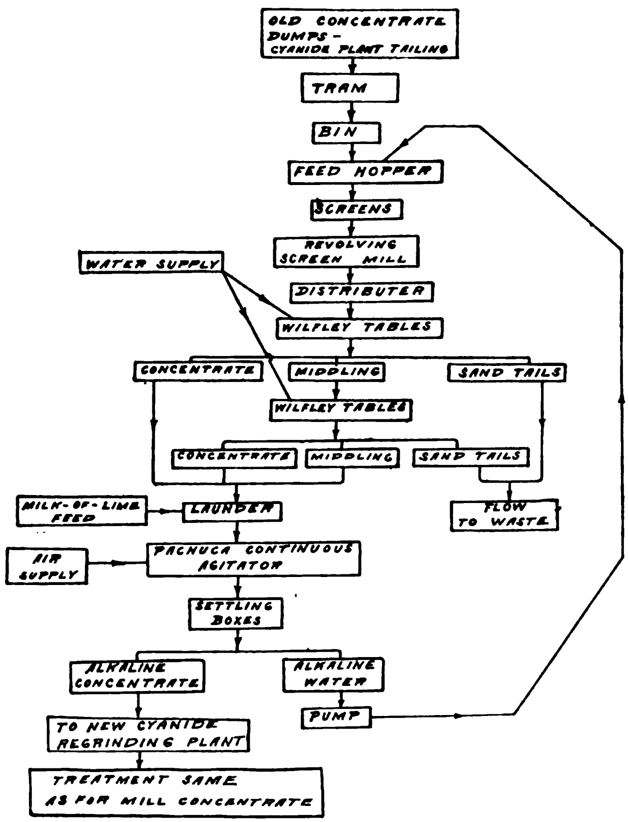


ORE TREATMENT CHART

this extraction, or even better, will be obtained. I am confident that the work of this plant will be fully up to expectations. The cost of this plant has exceeded the estimates, but it is expected to offset this by realizing a larger net profit than was originally estimated. The following are the results for the month of August, 1912, that were not at hand when the above report of the new

Taracol cyanide plant was completed. Total tons concentrate treated was 2000, of an assay value per ton of $29.14. This gives a total content of $58,289. The bullion obtained was $54,567, corresponding to an actual extraction of 93.6%, or a theoretical extraction of 90.1 per cent.

These results are better than were expected, and with cheap electric power to be supplied from Anchu to this plant the coming



DUMP RE-TREATMENT CHART.

year, the increased net profit ($20,880 per year over the results of the old leaching plant) originally estimated will be somewhat exceeded. Another re-treatment plant for the large concentrate cyanide tailing dumps at Taracol is now under construction. The process will be similar to that at the Kuk San Dong re-treatment plant. The accompanying dump re-treatment chart will serve to explain the process employed.

BLACK OAK MILL

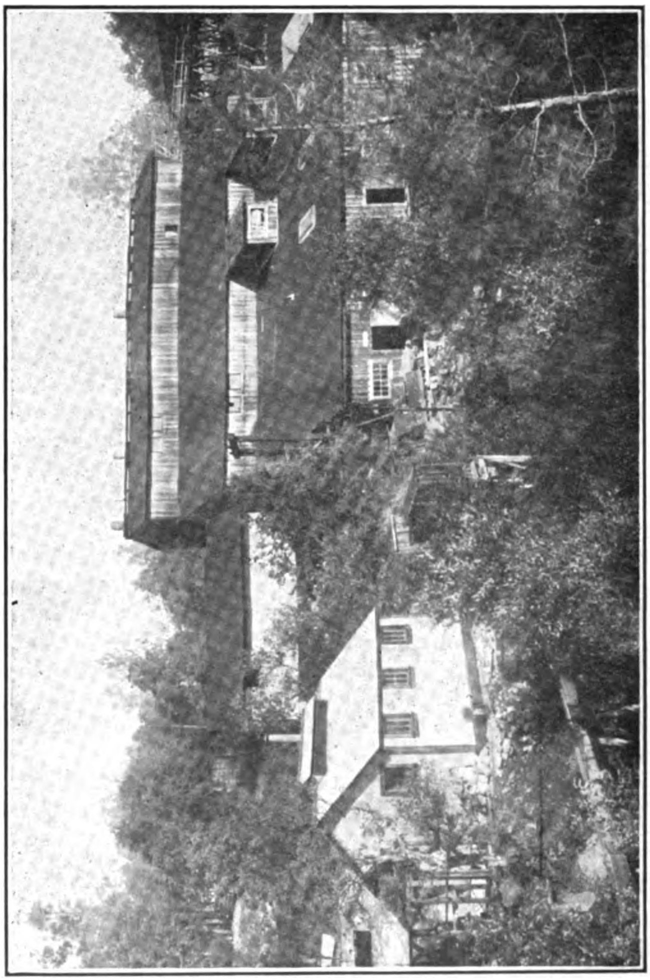
By Charles H. Urquhart

(November 30, 1912)

The Black Oak mill at Soulsbyville, on the East belt of the Mother Lode in Tuolumne county, California, marks a new departure in Mother Lode milling practice. It is, I believe, the first all-sliming plant in the State. As it has proved successful, a description may prove interesting.

The mine has been operated intermittently for many years and has an enviable record for gross production. The former method of milling was the customary crushing with light stamps, amalgamating, and concentrating with vanners and a canvas plant. The mill had 40 stamps. The extraction was never good, and many thousands of dollars' worth of high-grade tailing went down the creek. The present company quickly realized that more modern methods of recovering the gold must be employed if the mine was to be a financial success. From results obtained by cyaniding the concentrate, followed by careful tests with the ore, it was found that no unusual or insurmountable difficulties stood in the way of successful treating the ore by cyanidation. The present mill is the result. Nothing remains of the old mill but the building within which all the new equipment is placed, the feeders and the ore-bin.

The ore is fairly high-grade, is very hard and silicious, and requires much work to reduce it to pass a fine screen. Even when passing 200 mesh, it is still granular and not a slime in the accepted meaning of the term. It is roughly hand-sorted at the hoist, being dumped from the skip over a grizzly into a small bin holding but a few cars. The coarse material passes through chutes to the sorting platform and the sorted ore is crushed in a Blake crusher, after which it is trammed to the mill. Each car is weighed on platform scales and sampled before reaching the mill. The mill-bin is small, holding but 180 tons. This is not altogether a drawback, as it takes only a little time and labor to level off the ore and accurately measure the contents at any time. As a check on the daily tonnage crushed this has been found to be very valuable. The twenty stamps used in the old mill were light, 750 and 850 lb., and have now been replaced by new stamps weighing 1250 lb. The feeders are the Challenge, of the non-suspended type. The mortars are the high, rapid-discharge sort, with bases 12 in. thick, and weigh 8500 lb. each. The foundation for the mortars is a monolithic concrete block carefully constructed of selected quartz and granite. Screenings accumulated while coarse-crushing the quartz were mixed with clean river sand, thus making a sharp product for use in the concrete mixture. After shoveling of the concrete into the forms began, it never halted until the entire block was finished. After six month's continuous use no signs of deterioration have been discovered.

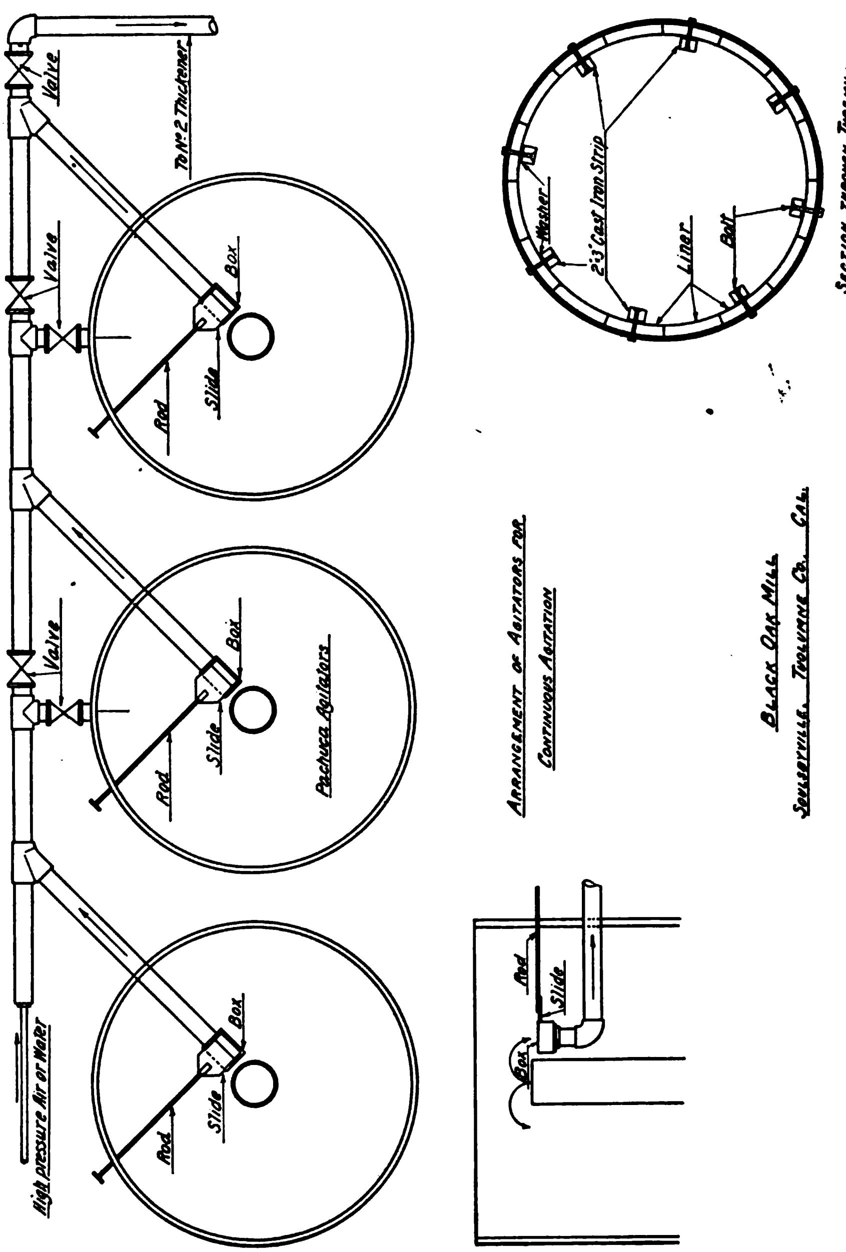


BLACK OAK MILL, SOULSBYVILLE, CALIFORNIA

The ore is crushed in 1.85% cyanide solution, or 3.7 lb. per ton, through Tyler ton-cap screens with openings equivalent to 10 mesh. The pulp passes to a Dorr standard duplex classifier from which the coarse material goes direct to the 5 by 18-ft. tube-mill, being diluted to 39% moisture. The slime-overflow passes to the treatment-tanks. The discharge from the tube-mill is returned to the classifier by a bucket elevator, so that the pulp must return to the tube-mill until fine enough to escape at the slime end of the classifier; the usual closed-circuit arrangement. A Morse No. 25A 6-in. silent chain and spring sprocket drive takes the place of the customary belts and countershafting. It makes a very neat and compact arrangement, the centres of the motor and tube-mill pinion shafts being but 42 in. apart. Absolutely no trouble from this drive has been encountered during the seven months that the mill has been operating, and it is evident that a considerable saving in power has been made over the belting and counter shaft method of driving.

Considerable trouble was had with the lower bearings of the bucket elevator. Sand and slime worked into the bearings and rapidly destroyed them. This was overcome by placing the bearings 7 in. from the elevator, outside the boot, and putting collars on the shaft between these outboard bearings and the boot to act as baffles. The bearings now keep quite clean and show no wear.

The liners of the tube-mill are smooth cast iron, a mixture of chrome, manganese, and white iron; and are made in a local foundry. They are very hard and, judging from the present rate of wear, should last about twelve months. They are 13½ in. wide, 41 in. long, and 2 in. thick, set staggered with shorter pieces at the ends. Lugs or pads back of the bolt-holes, 4 in. square and ¼ in. thick, prevent breaking the liners when bolting. No cement is used, as the narrow space behind the liners soon fills with slime. After the mill had been running a few weeks it became apparent that the tube-mill was not keeping up to its early performance. The pebbles were wearing flat and the inference was that they were skidding in the mill, thus losing the tumbling action requisite for effective grinding of this hard ore. Cast strips of hard iron were procured, 3 in. wide, 2 in. thick, and of the same length as the liners. These were bolted in the tube in such a manner that they made long strips 2 in. high the entire length of the tube; a strip for every alternate row of liners. Bolts passing through the strips and main liners take the place of the original bolts; a washer of old belting between the strips and the liner preventing the hard brittle strips from breaking when tightening the bolts. The results were marked, and the output was increased at once. Again the tube-mill was opened and another set of strips put in, now making a strip for every row of liners. The grinding at once fell off and the starting-load of the motor became excessive. The second set of strips was taken out and the efficiency of the mill returned to the point first reached by the original arrangement of strips. Since these strips have been employed the output of the mill has been increased 13.4%, and the life of the main liners has been materially lengthened. After four


To No. 2 Thickener
Valve
Valve
Valve
Box
Slide
Rod
Box
Slide
Rod
Pachuca Agitators
Box
Slide
Rod
High pressure Air or Water
2'3' Cast Iron Strip
Washer
Liner
Bolt
Section through Tube
Arrangement of Agitators for Continuous Agitation
Black Oak Mill
Soulsbyville, Tuolumne Co. Cal.
Rod
Slide
Box

months' steady work the strips show but a slight rounding of the upper edges. The mill revolves 23 r.p.m., taking practically 60 hp. to operate and 100% overload to start.

The pulp from the classifier goes to a 20 by 10-ft. Dorr thickener, where it is thickened from a 6 to 1 consistence to 2 to 1 for agitation. When the mill was started the pulp from the thickener was elevated to the agitators by a centrifugal pump, but in a very short time the hard gritty slime abraded the liners, flanges, and impeller to a ruinous degree. A 2½-in. air-lift was then substituted, and although the work to be done was not in accordance with good air-lift practice, the total lift being 18 ft. and the submergence but 50%, nevertheless the results have been quite satisfactory and no repairs nor alterations have been necessary. The power consumed is 6 horse-power.

The agitators are three in number and of the Pachuca type, 10 by 30 ft., and connected for continuous agitation. They are so arranged that any agitator can be by-passed and cut out of the circuit, as illustrated in the sketch. In order to make certain that the pulp of uniform grade passes continuously through each agitator, a very simple sampling box is arranged on the end of the pipes connecting them. On the top of the box is a slide which is readily adjusted so as to regulate the amount of pulp flowing in and out of each agitator. This takes an accurate sample of the pulp as it falls from the central tube; a very essential matter when treating a pulp which segregates as rapidly as does the granular product treated here. A spider with six radiating jets supplements the main working air jet as a help in starting at any time. The spider and the main jets are connected with both high-pressure aid and high-pressure water, although the water never has been required. An Ingersoll-Rand 8 by 10-in. compressor furnishes air at 40 lb. pressure for the agitators, the air-lift, and the vacuum-filter. The power consumed for agitation as shown by the meter is 3.4 hp. per agitator.

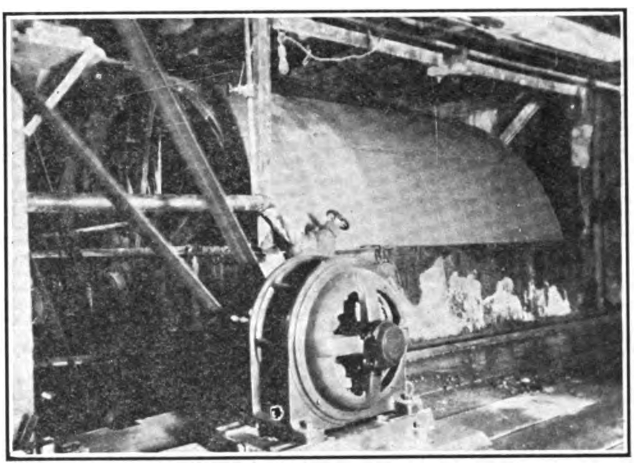
Having passed through the agitators the pulp flows to another Dorr thickener, the same size as the first. The overflow from No. 2 thickener goes to a small collecting-tank, as it is called, from which it is pumped to the clarification and precipitation presses. The thickened underflow passes to the filter. As the ore milled averages $20 per ton, the resulting solutions are high in value. In order to reduce the value of the solution in the pulp going to the filter, the contents of the No. 2 thickener are diluted as follows: The overflow from the No. 1 thickener, assaying about $3 per ton, is passed to the centre of the No. 2 thickener, mixing with the inflowing pulp coming from the agitators, containing solution which assays around $8. As there is twice as much solution coming from the No. 1 thickener as from the agitators, the resulting mixture of solutions in No. 2 has thus been diluted to about $3.30 per ton in value. To further dilute this solution, barren solution from the stock tanks is run into the second thickener inflow and also to the thickened underflow. Thus the solution that came from the agitators worth $8

per ton has been reduced by dilution to under $2. The filter is an Oliver with a drum 11 feet 6 inches in diameter and 12-ft. face. It is rated by the makers as a 75-ton machine, but with our ore it can easily take care of 100 tons per day. Considering the many excellent filters on the market, the problem of selecting the best one for a new mill is not an easy one. At the Black Oak the Oliver was chosen because in theory the washing of a thin cake seemed more practicable than the washing of a thick cake; because the filter was continuous, simple, and cheap to install and operate; and finally, because a special filterman was not required. Now that the mill crew has become familiar with the machine, it has been found that these conclusions have been borne out in practice and the filter has proved to be splendidly efficient. In the beginning poor results were obtained with the washing. The cake was ½ to ⅝ in. thick and the displacement was incomplete, as much as 20c. being lost at times in dissolved gold and cyanide with the tailing. A spraying device was later supplied by the Oliver people, and the difficulty was solved at once. The pulp-level in the tank is kept fairly low and an air wash given until the cake emerging from the pulp in the tank has traveled half the distance, when the line of sprays is met and the washing begins. Since the sprays have been installed the displacement has been almost perfect; the solutions are easily kept in accurate balance, and no excess has been built up by over-washing. The filter takes but a small proportion of the solution-man's time, and the power consumption is small. Repairs are infrequent and cost but little.

Soon after the plant was started a curious black deposit began to form on the filter cloth. It spread rapidly, clogging the cloth, and reducing the available filtering area almost one-half. Hard scrubbing with a 6% HCl solution twice a week eventually removed the deposit. Later the scrubbings were made but once a week, and the HCl solution reduced to 4%. The cloth has been on for seven months and, barring accidents, should last many months to come. The weekly scrubbings are doubtless the secret of the long life. The filter is scrubbed once a week whether the cloth apperas to need it or not. The Oliver filter requires a vacuum pump if much higher afficiency and greater capacity than a leaf type of filter of similar capacity, not only because it is essential to maintain a higher vacuum than is used on the leaf filter, but because a larger quantity of air has to be handled, due to the exposed portion of the drum. For these reasons the types of vacuum pumps commonly in use for leaf-filter plants give very poor results with the Oliver filter.

The pump installed at the Black Oak mill was built by the Doak Gas Engine Co., being a modified type of a pump especially designed by George D. Kislingsbury, superintendent of Minas del Tajo, Rosario, Mexico, for the three Oliver filters there. It has a 12 by 10-in. cylinder, and differs from other wet-vacuum pumps in that there are no inlet valves, there is practically no clearance, and the discharge valves are in the piston.

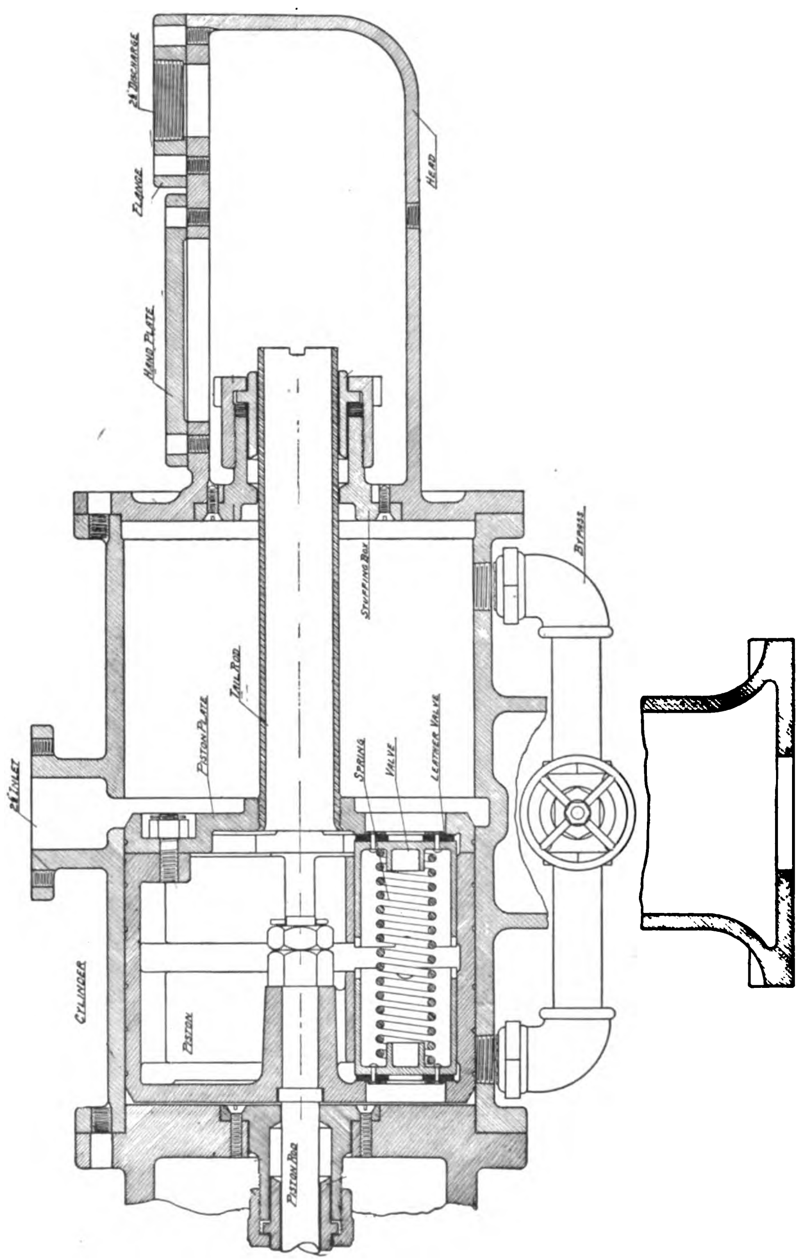
The solution is admitted to the cylinder through ports in its centre, which are opened and closed by the passage of the piston. The solution is discharged through a hollow tail-rod into an enclosed false-head which serves the double purpose of allowing the solution te be discharged under pressure and acting as an air chamber to relieve the shock of the pump. Because of the unusually large valve area the pump may be operated at a speed of 125 r.p.m. without shock, and the small clearance space admits of discharging under relatively high heads without loss of efficiency. A peculiar feature of this pump which puzzles most operators is that it requires only half as much power to operate at 25-in. vacuum as at 12-in. vacuum. The actual power consumed while operating the filter at 22-in. vacuum is 6 horse-power.



OLIVER FILTER, BLACK OAK MILL

The solution from the filter discharges into the collecting-tank mentioned above and is pumped to either of two 24-in. 36-frame Shriver filter-presses for clarification. From the gold-tanks that receive the clarified solution it is pumped to a 52-in. 16-frame Merrill press, zinc dust being added in the customary manner to the suction of the pump. While no difficulty has been found in precipitating the gold almost completely, by allowing the effluent to assay 10 to 15c. per ton, the tendency to accumulate excess zinc is avoided.

The barren solution passes from the precipitation press to the sump, from which it is returned to the stock tanks above the mill by a triplex 5½ by 8-in. Platt Iron Works pump. It is there brought back to normal strength. Lime is added to the batteries and in the agitators. As the quality of the lime varies, the alkalinity of the solutions is determined in terms of percentage of free



WET-VACUUM PUMP CYLINDER FOR T-O TYPE

alkali rather than in pounds per ton. The amount added by the solution-man therefore varies slightly from day to day, the working percentage being 0.15% CaO.

Until quite lately lead acetate, 30 to 40 lb. per day, was added with the cyanide in the stock tanks and in the agitators, but it was found that 24 lb. of commercial litharge would do the same work. As this was considerably cheaper the use of lead acetate was dropped and the litharge substituted. It is added in the tube-mill and in the agitators; an equal amount in each.

A car with a perforated false iron bottom receives the precipitate when cleaning the precipitate press. When the press is empty the car is taken to the refinery, a separate building 75 ft. from the mill. Compressed air which has been heated by passing it through a coil in a small furnace is introduced beneath the false-bottom of the car; as it proceeds up through the mass it rapidly dries the precipitate. It is weighed on platform scales, fluxed, and smelted in a No. 2 Rockwell double-chamber furnace, no previous acid treatment being necessary. The resulting bars of bullion are base, averaging 850 fine, and a change of the refining method is under consideration. This will possibly include the use of an English cupelling furnace.

High-pressure air from the mine compressor is piped to the mill, acting as as auxiliary in case of emergency. If for any reason the agitators stop and the pulp settles so that difficulty is found in resuming agitation, the high-pressure is turned on. It is also piped throughout the mill, so that the various motors can be blown free from dust at regular intervals. It is connected to the filter-presses and to the refinery for the crude-oil burners. The main floor of the mill is cemented and provided with drains that all lead to the sump, so that all leaks or overflows are caught.

The mill is equipped with Mazda 'Street Series' tungsten lamps and is brilliantly lighted at night. For mill illumination these lamps are very satisfactory. They have extra heavy filaments, take but little current, cost but a nominal sum to replace, and when suspended with light springs have a long life. They have proved so satisfactory in the mill that the company has installed them over the surface workings and at the hoist and ore-sorting platform. The mill is operated entirely by electric power, which is purchased from the San Francisco & Sierra Power Co. Coming in at 15,000 volts, it is transformed to 440 for power purposes, and further stepped down to 110 volts for lighting.

Since the mill started, the average monthly extraction has never fallen below 95%, and for the last three months has been over 97%. This is the total extraction of gold and silver, and is the actual money shown by bullion returns, not the 'indicated.' The returns on bullion check very closely with the indicated or calculated extraction of the daily mill estimates, usually slightly exceeding them. The cyanide consumption averages 1.5 lb. per ton of ore; the lime 3 lb. One battery-man and a solution-man per

shift run the mill with ease. A mechanic attends to the repairs on the day shift, and the mine assayer and a boy do the smelting. The daily capacity of the mill, using light stamps, was 66 tons, but with the new 1250-lb. stamps the output should be at least 100 tons per day with no increase in labor.

OPHIR CYANIDATION PLANT

(November 30, 1912)

The new plant of the Ophir Silver Mining Co., operating on the Comstock Lode, has now been in operation a little over a month, and was briefly described in the *Virginia City Chronicle*, by Walter Techow, who, after designing and erecting the mill, is now in charge of its operation. In building the plant an excellent record for speed was made. It was ordered by the directors June 22. The first work of grading began on July 15, and while the lumber did not arrive on the ground until the end of the month, the actual construction began early in August. The frame was put up during that month, and the building inclosed early in September. The machinery was also installed in September, the finishing touches made early in October, the plant started October 14, and has been running continuously ever since. The plant is complete with the exception of the clean-up room, now being added, and soon to be placed in commission. With the melting facilities available, the first clean-up will be made.

According to Mr. Techow, the plant was designed for a capacity of 100 tons per day. During the first two weeks after starting, it treated 1065 tons. During the third week 625 tons was handled, and during the past week the plant has been running at a rate of more than 100 tons per day. The process adopted consists of re-grinding the coarse part of the tailing from the old Kinkead mill, agitating the re-ground tailing with cyanide solution, and separation of the gold and silver-bearing cyanide solution from the now barren tailing. The tailing is re-ground in a 5 by 22-ft. tube-mill furnished by the Union Iron Works. One problem in connection with the erection of this mill was how to get the wet and sticky material to the tube-mill, and the method adopted was to sluice it from the pond to the tube-mill. This method has proved successful and economical, as the cost of getting the tailing into the mill is only about 9c. per ton. The tailing is sluiced into the tube-mill with dilute cyanide solution and is continually in contact with cyanide solution from the moment it leaves the pond until the treatment is completed.

From the tube-mill the re-ground tailing passes to a Dorr thickener 28 ft. in diameter by 8 ft. high, where it is settled and the excess of the cyanide solution is removed. This solution is pumped back to the tailing pond and is kept continually in circulation. The settled slime is elevated from the bottom of the Dorr thickener into one of a series of three agitating-tanks, 16 ft. high

and 22 ft. in diameter. These agitating-tanks are equipped with Trent agitators. They require little power and do excellent work if properly erected and looked after. After about 24 to 36 hours agitation with an additional amount of cyanide, the slime is transferred to a storage tank and passes from this tank to a Butters filter. The gold and silver-bearing cyanide solution leaves the filter clear, and passes on to the precipitating-boxes, while the tailing is discharged from the Butters filter and allowed to run to waste.

The original estimate of the cost of operation of this plant was $1.25 per ton. It appears now, however, after one month's operation, that the estimate was too high. The cost of treatment will probably not exceed $1.20 per ton, and possibly may be lower. The consumption of cyanide is at present less than three-quarters of a pound per ton or an equivalent of $0.19 per ton. The total cost of labor will come to about $0.40 per ton. The power consumed during the first two weeks was at the rate of 100 hp. It will probably be less in the future, as all the machinery was new and in the process of being limbered up.

Up to the present time, by the records at the plant, the tailing treated has averaged about $5 per ton before treatment and $1 per ton after, thus indicating an extraction of approximately 80%. It is believed, however, that further experiments and improvements will raise the extraction to 85%, or possibly higher. It will be of interest to again recall that the tailing is the product of the Kinkead mill, which by the process of concentration, extracts 84 to 87% of the original value of the ore as shown by the smelter bullion returns. It can thus be concluded that a final loss of $1 per ton represents only 2 to 3% of the first head sample at the Kinkead mill. The total extraction at the mill and cyanide plant will, therefore, undoubtedly average 97% or better of the gross value of the ore. This extraction, according to figures furnished at the Ophir office, is being obtained at a total cost of $3.75 per ton for both milling and cyaniding.

ROSARIO CYANIDE PLANT

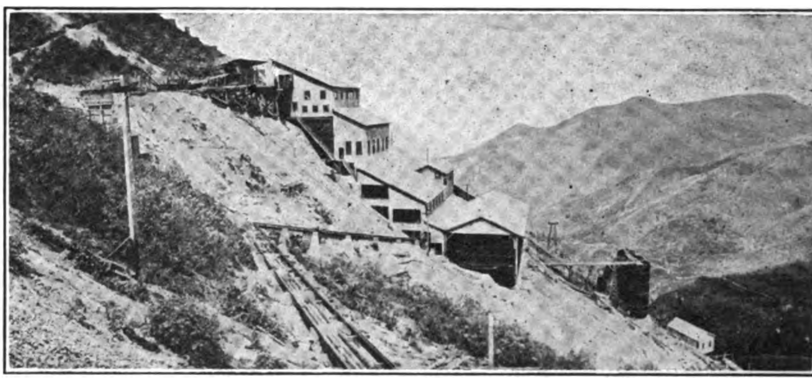
By A. L. Sweetser

(December 14, 1912)

Twenty-five miles east of Tegucigalpha, capital of the republic of Honduras, and two miles west of the town of San Jirancito, is situated the surface equipment of the New Yory Honduras Rosario Mining Co., which is the largest producer in Honduras, if not in Central America. The property may be reached either from Puerto Cortez, on the Gulf of Mexico, by a six-day trip over mountain trails on muleback, or from Amalpa over a wagon road requiring four days for a saddle animal and about three weeks for heavy freight. Considering the isolated situation of the

mine and the difficult problem of freighting heavy machinery to it, the following brief description of a cyanide plant recently constructed there should prove of interest.

The site selected for the erection of the plant has a slope of 37°, thus facilitating the transfer of material by gravity.. The foundations consist of concrete piers which are sunk in the hillsides to a depth of 20 ft. and securely anchored to bedrock. In order not to disturb the natural condition of the loose gravel above the bedrock, all the holes for the foundation piers were not dug at the same time. For instance, where a building required two or three rows of five piers each, the order in which these holes were excavated and then filled with concrete was 1—4—2—5—3. This method prevented the loose surface material from sliding and causing the disturbed gravel to creep, as might have occurred were all the holes excavated at the same time. On these concrete piers



ROSARIO CYANIDE PLANT, SAN JUANCITO, HONDURAS

steel I-beams were laid, on which were placed the buildings, consisting of board floors and corrugated iron sides and roofs.

Wherever the building required a large floor space, the gravel of the hillsides was cut down and leveled. Then retaining walls of concrete were built along the back and sides. The angle of these walls is 67°, and they have a thickness of 4 ft. at the bottom, but taper off to 1 ft. at the top. The back wall is also reinforced by concrete braces, built at an angle of 35°. These vary in height and thickness, according to the height of the retaining wall. The composition of the concrete used throughout the construction was 1 cement, 3 sand, and 5 parts of stone. All the concrete was reinforced, during the pouring, by the insertion of old mine rails, pieces of cable, and scrap iron. The total consumption of cement in the construction was 15,000 barrels.

In addition to the mill, there are the following buildings: A general office which, in addition to other modern equipment, contains a printing plant for the production of the daily report sheets

for the various departments and general notices; a large storehouse and assay office, with latest appliances for facilitating rapid and accurate work; mechanical, electrical, and carpenter shops, and houses for the storage of lime, cyanide, and zinc. A central telegraph and telephone station connects all departments of the mine and mill with the general office, and also permits communication with all towns toward Puerto Cortez, on the Gulf of Mexico, and Amapala, on the Pacific, over a hundred miles distant. Dwelling houses, with well ventilated rooms, electric lights, toilet, and shower baths, and a large dining hall complete the number of buildings. A small ammonia ice-plant enables the cooks to serve some delicacies not often found in a mining boarding house. All the vegetables come from the company's farm, and all the lumber used in constructing the buildings and in the mine is produced in the company's sawmill from timber from the forest, 9 miles distant, also belonging to the company.

Cyanide Plant

The general exterior arrangement of the plant can readily be seen from the preceding photograph of the same, and the interior arrangement of the ore dressing machinery is shown by the flow-sheet.

Flow-Sheet

1. Fairbanks-Morse scale.
2. Tipple.
3. Grizzly.
4. 2 Gates crushers, type K 4.
5. 1000-ton ore-bin.
6. 20 1850-lb. stamps; four batteries of five each.
7. 2 Dorr classifiers.
8. 2 Allis-Chalmers tube-mills, 22 by 5 feet.
9. 1 Dorr classifier.
10. 1 tube-mill to complete sliming of pulp.
11. 2 settling tanks, 10 by 20 feet.
12. Screen on which cyanide and lead acetate are dissolved by solution.
13. Pachuca tanks, 15 by 45 feet.
14. 2 storage tanks, 10 by 20 feet.
15. 2 Merrill filter presses; capacity 220 tons.
16. 2 strong and weak-solution tanks, 10 by 15 feet.
17. Zinc dust precipitation.
18. Refinery.

Operation

The ore, as delivered at the mill, averages about 25 oz. silver and 0.46 oz. gold per ton. In addition to these metals the ore also contains copper, lead, iron, and antimony in small quantities. The stamps are in separate battery units, so that when it is

necessary to make alterations, or repairs, only one battery of five stamps is hung up at a time. Each stamp weighs 1850 lb., drops 100 times per minute with a fall of 6 in. The stamp-duty is 10 tons per stamp per day, thus making the mill capacity of 200 tons daily, although an excess of 20 tons has frequently been treated. The ore is mixed with lime, in the bin, by means of wooden chutes from the crusher floor. Each battery is fed by Challenge feeders and the ore is crushed in the strong cyanide solution from the Merrill presses, which is pumped from the precipitation house by means of two Aldrich triplex pumps (10 by 10 in.) to storage tanks above the mill, whence it flows by gravity. The weak solution, similarly pumped, is used as a wash for the Merrill presses.

On leaving the settling-tanks the solution and pulp flow over a screen on which is placed the required amount of cyanide and 50 lb. of lead acetate. From repeated tests during the six months the mill has been in operation an average of 500 lb. of cyanide is required for each Pachuca tank of pulp. The pulp in the Pachuca tanks is agitated by means of compressed air supplied by an Ingersoll-Rand duplex compressor, 5 cu. ft. of air being furnished to each tank. The amount of sluicing water, used to remove the cakes from the filters, per ton of dry slime treated is about 4½ tons; the ratio of wash solution per dry ton of slime is 2.9 to 1.

Precipitation is accomplished by means of zinc dust agitated by compressed air. An extraction of 91 to 96% is obtained and the difference between the theoretical and the actual extraction is less than 1%. The press precipitate is melted into bars which average in weight about 125 lb. These are shipped to Perth Amboy, New Jersey.

Electric Power

All of the mill machinery is motor-driven. General Electric induction motors, 60 cycles, are principally used. The Gates crushers are driven by 25 hp. form K motors, using 6.6 amperes and a 75-hp. form M motor, using 20.5 amperes, drives each tube-mill. For the pumps 25-hp. motors are used, 2 hp. for the Dorr classifiers, and 15 hp. for the filter-presses.

The present system of electric power dates back ten years, when the first electric plant was installed. From this time the electrification of both mine and mills has proceeded rapidly, until now there are two hydro-electric generating stations, furnishing 6600 volts and supplying 50 motors, having an aggregate capacity of 2100 horse-power.

The water, furnishing the initial power, is taken from the San Juancito and Escobales rivers, and conducted in a wooden flume 3½ miles to a point 1355 ft. above the town of San Juancito, whence it descends in a spiral-riveted, iron pen-stock, made with expansion joints to provide for changes in atmospheric temperature. The water pressure of 520 lb. per square inch acts on Pelton water direct connected to 6600-volt generators. The Guadalupe generating station is three miles below the town and utilizes the water

escaping from the San Juancito spillway. Each station can, if necessary, carry the entire working load at a slightly diminished efficiency. The service is practically continuous, there being a demand on the power system at all hours. Electric power is transmitted to the sub-station at the mill, 2 miles distant, whence it is distributed throughout the mill, mine, repair-shops, assay office, and lighting system.

The average demand of the mill is approximately 450 hp. and that of the mine 600 hp. At the mine there are two 800-hp. air-compressors, producing 2000 cu. ft. of air for drilling purposes, and also six 5-ton General Electric mine locomotives which gather the ore on the different levels of the mine. It is sent to the adit-level and later to the mill, two miles distant. The mine has been in operation over thirty years and the underground workings now amount to 60 miles. The trackage of 35-lb. rails and 2½-ft. gauge is being laid through the mine as rapidly as possible to facilitate the transportation of ore to the mill and the workmen to the various stopes.

All the machinery in the mill is direct connected to motors. The stamp units are belt-connected to a counter shaft driven by motors, which are in duplicate in case one should be out of commission. All wiring is in iron conduits and the lighting service at the mine, mill, on the surface, and in other buildings consists of 800 Mazda incandescent lamps and 25 arc lights. On the battery-floor of the mill carbon filament incandescent lamps are used, because the vibration due to the impact of the stamps causes a large breakage in the case of metallic filament lamps.

TOMBOY MILL

(December 28, 1912)

This mill consists of 60 stamps and a concentrating and cyanide plant. During the year ended June 30, improvements were made and the following details are published in the recently issued annual report. The grading plant, consisting of three Wilfley tables and one Diester slimer, receives the mixed concentrate of the Wilfley tables, and the entire concentrated product of the Frue vanners, and turns out a finished product of high-grade galena concentrate, over 60% Pb, carrying much gold; a zinc concentrate containing 24% Zn, and higher; and an iron product which, under the present smelter schedule, commands a much more favorable price for gold, silver, and lead than when, formerly, the iron was included in the zinc concentrate.

Previous to remodeling of the concentrating room, the treatment of the entire mill product was accomplished by means of 15 Wilfley tables and 12 Frue vanners. Of the former, four were restricted to re-treatment of middling and concentrate, leaving 11 for the actual work of primary tables. The construction of the grading plant for re-treatment of concentrate liberated two Wil-

fleys, while the sending of the middling to the middling plant released two more Wilfleys for the concentration of original feed. The addition of two new Wilfleys on the north end of the mill, three on the south end, and the installation of two Wilfley roughing tables, which materially relieve the work of the other tables, makes twenty-two tables working on primary feed.

The middling plant was designed by Gelasio Caetani for the re-treatment of the high-grade tailing, which repeated tests proved to be contained in that portion of the tailing nearest the middling streak on the Wilfley tables. The feed to this plant will be obtained from the higher grade portion of the tailing discharged from the Wilfley roughing tables, and the high-grade tailing from the primary Wilfleys. The plant consists of two Hardinge conical tube-mills for re-grinding, eight copper plates for re-amalgamation, classifiers, seven Callow tanks, seven Deister slimers, and eight Wilfley tables for the re-concentration of the re-ground middling product. For the relief of the filter plant this year, and with a view to its entire displacement in the future, it has been decided to employ four 10 by 32-ft. Dorr thickeners. It is hoped that these improvements in the recovery of clear water will permit of the maintenance of increased tonnage due to new plant throughout the year.

During the year ended June 30, 1912, the mill treated 107,577 tons, yielding $954,981, at the following cost:

Milling, per ton	$0.67
Concentrating, per ton	1.00
Water supply, per ton	0.19
Assaying, per ton	0.04
General and taxes, per ton	0.59
Total, per ton	$2.49

MACNAMARA MILL, TONOPAH

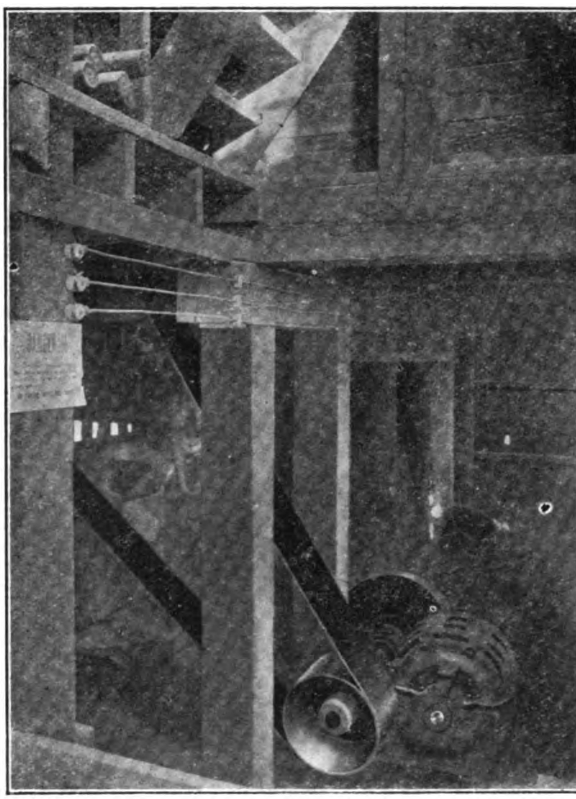
By M. W. von Bernewitz

(January 25, 1913)

This modern little plant has been in operation for a year and is satisfactorily treating a hard ore carrying 20 oz. silver and a few grains of gold per ton. An electric hoist, with flat ropes, at the main shaft hauls self-dumping skips of 2000-lb. capacity to the surface, where the ore is emptied into a storage-bin. The ore is fed upon a 36-in. rubber sorting-belt; about 6% is discarded as waste, the remainder then passing through a No. 3 Kennedy gyratory crusher, and is taken to the mill-bin by bucket-elevator. This bin has a flat bottom, is of 200 tons' capacity, and has the ordinary rack-and-pinion gate on the ore-chutes to the feeders.

The mill includes ten 1400-lb. stamps, dropping 8 in., 98 times per minute, and crushing 7 tons per stamp per day, through a No.

12 ton-cap slotted screen. As will be seen from the accompanying illustration, each 5-stamp battery is driven by a 20-hp. Westinghouse back-geared motor. This may be described as a type 'M.S.', 440-volt, 60-cycle, 3-phase motor, running at 555 r.p.m., the pinion on the armature shaft engaging with spur wheel on the short pulley-shaft which runs in bearings on the back of the motor, there being a 14-in. belt driving the bull-wheel on the 6½-in. cam-shaft. These motors have given complete satisfaction, and they were



BACK-GEARED MOTOR DRIVING STAMPS

recently opened after about eight months' work, inspection showing that the tool marks were hardly worn off the gear-wheels, which work in grease. No doubt this style of drive reduces friction greatly, as there is no counter-shaft, except the little pulley-shaft, and no tightening gear for the belt. The friction in this case amounts to about 9%. Careful tests have shown that five stamps absorb 17.75 hp., including 1.61 hp. for friction, or 3.55 hp. per stamp. As a check on this reading, the following old rule can be used:

$$\frac{1400 \times 98 \times 8}{33{,}000 \times 12} = 2.77 + 20\% \text{ for friction} = 3.32 \text{ hp.}$$

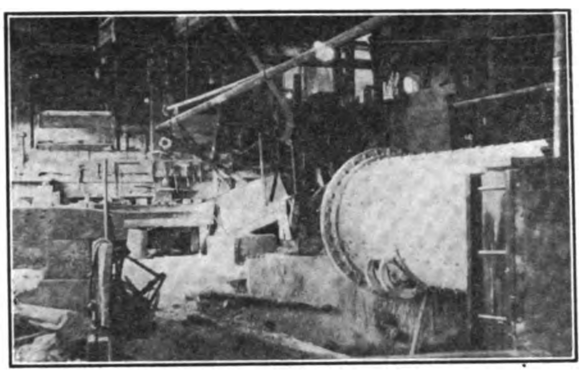
It may be added that at the new Belmont 60-stamp mill there is a 60-hp. motor for 20 stamps, which weigh 1250 lb. each. This drives a short jack-shaft, which in turn drives two batteries of 10 stamps. This is similar to the drive at the Goldfield Consolidated, only there the motors are 50 hp. each, and at the Desert mill of the Tonopah Mining Co. the same is true. The West End mill has four units of five stamps each, actuated from a common motor-driven counter-shaft by friction clutch-pulleys, which arrangement gives no trouble. Besides the two motors driving the MacNamara stamps, there is a back-geared motor driving a Dorr classifier and agitator in the thickening-tank. This method of transmitting power is well worth the attention of all engineers.

Foundations for the two batteries are of reinforced concrete. The mortars (Chalmers & Williams, No. 16-E) have a heavy base, and stand on sheets of rubber; while the battery-posts are set in cast-iron sole-plates, also standing on rubber. The cam-shafts are 6½ in. diam. The ordinary Blanton cam is used, and none have been broken so far. Stems are 4 in. diam., working in Pacific guides made by the Demarest company, and none have been broken. The Pacific guide consists of a metal frame into which are fitted cylindrical metal shells that are not held by keys or the like. The screens are fitted 3½ in. from the stamp-shoes, and through a No. 12 ton-cap square-slot screen the average duty per stamp was 7 tons of ore per day, but recently coarses screens have been used, and the duty has risen to over 8 tons. Crushing is done in weak cyanide solution having a temperature of about 70°F. Feeders of the improved Challenge type, which may be called a double monkey-wrench grab, deliver the ore to the stamps.

The pulp from the stamps goes direct to a Dorr classifier, working at 12 strokes per minute, the coarse material being fed into a 5 by 16-ft. tube-mill revolving at 26 r.p.m., through a spiral feeder. Pebbles are fed in at this point, amounting to 250 lb. per day, consumption being about 3½ lb. per ton milled. Slacked lime equal to 2½ lb. per ton is added in the classifier. The tube-mill discharge is returned to the classifier by a 5-ft. Frenier pump, which gives no trouble, the coarse material again passes through the tube-mill and so on, forming the usual closed circuit, in which only the overflow from the classifier can get away. In the batteries, 25% of the ore is slimed, while the final pulp, which is pumped to a Dorr thickener, shows 73% through 200-mesh screen. This is fairly coarse when compared with other mills, but is found to be fine enough for good results.

All the classifier overflow is pumped to a Dorr thickener, 12 by 26 ft., the gear of which travels at one revolution in eight minutes. Clear solution overflows to the battery storage-tanks, and when its silver content becomes high, it is decanted off to

the 'silver' tank. The thickened slime, specific gravity 1.22, flows by gravity to three 15½ by 25½-ft. Trent agitators for 48 hours' agitation in a 2-lb. KCN solution. The agitators hold 80 tons of dry slime each, and cyanide and lead acetate are added here, as well as live steam, bringing the temperature up to 115 to 120°F. This has been found beneficial, as with cold solutions extraction falls off considerably. This type of agitator gives satisfaction, and even after an enforced shut-down, little trouble is experienced in starting. Air for the agitators is produced by a motor-driven compressor with 6 by 10-in. cylinders, the working pressure being 25 lb. per square inch. Each agitator has a 4-in. Campbell & Kelly centrifugal pump, direct-driven by a 7½-hp. motor, for circulating the slime through the agitator arms. From the Trent vats, the slime is pumped to a 19½ by 27½-ft. stock or storage-tank, which has a chain-driven pump for circulation.



STAMPS, CLASSIFIER, AND TUBE-MILL

From the stock tank, pulp flows by gravity to the vacuum plant, which consists of 50 leaves, each 5 by 10 ft., and treating 20 tons per charge. Forming cakes takes one hour, and washing two hours, with a 5-minute final water-wash at times, while the whole cycle consumes about four hours. The cloths last well, and have been on about 12 months so far. They are given a 24-hr. bath in 2% hydrochloric acid, about two leaves being treated per day. The washed slime-cakes are blown off with high-pressure water into a full tank of water, and the pulp pumped away by a 4-in. pump to ponds. Although the slime from the storage tank goes to the filter-vat by gravity, it is helped by a belt-driven Campbell & Kelly pump, which also pumps excess back to that tank, and does the circulating in the filter-tank. In October last the filter-tank was struck by lightning and completely destroyed, save the leaves; but in a short time a new one was built.

Solutions from the filters are clarified and run through four precipitating-boxes charged with zinc shavings. The precipitate

is dried to about 10% moisture in a steam-drying pan, mixed with fluxes, and melted in an oil-burning tilting-furnace. An extraction of 93% is made. The operating cost per ton in November 1912 was as follows:

Crushing and conveying	$0.176	Assaying	$0.071
Stamp batteries	0.340	Superintendence and foreman	0.233
Door classifier	0.145	Heating solutions	0.253
Tube-mill	0.523	Water service	0.178
Door-thickener	0.030	Compressed air	0.017
Agitation slime	0.776	General expense	0.073
Filtering and discharging slime	0.237	Surface and plant	0.050
Precipitation	0.116		
Refining	0.097	Total cost	$3.299

A JOSHUA PALM. A BIT OF THE NEVADA DESERT

REVIEWS OF PROGRESS BY YEARS

PROGRESS IN TREATMENT OF GOLD AND SILVER ORES DURING 1910

By Alfred James

(January 7, 1911)

General.—It seems particularly appropriate that America should this year take pride of place in progress. During past years it has been customary to refer to the practice at Kalgoorlie, or New Zealand, or South Africa, as ranking highest. This year improvements effected by such men as F. C. Brown (Idaho), J. V. N. Dorr (Denver), A. Grothe (Mexico), and C. W. Merrill (San Francisco), have been more generally adopted, and it can scarcely be doubted that for advanced successful practice such mills as Philip Argall's at Stratton's Independence, Goldfield Consolidated, Homestake, Esperanza, San Rafael, Dos Estrellas, and El Oro, take precedence. Kalgoorlie, renowned some years back for its successful pioneer work in the treatment of telluride ore, appears lately to have been resting, and its main effort for the last two years would seem to have been the adoption in a few cases, mainly for re-treatment purposes, of the fixed submerged (Cassel) filter. But this position of calm by no means indicates that all difficulties have been overcome. On the contrary, metallurgical troubles are possibly more prominent in the Colony now than for some years past. One misses the battle-cries of the stalwarts, 'Roasting *v.* Bromo-Cyanide', 'West-Crushing *v.* Dry-Crushing', 'Tube-Mills *v.* Pans'. Since Robert Allen published his valuable work on Western Australia practice but little new has come to light except for an occasional thought-compelling article by M. W. von Bernewitz, whose contributions to current technical literature should surely be collected for future reference.

Africa still pursues its determined course of increasing the weight of stamps, although sound unshaken criticism shows no gain from such increased weight other than resulting from the employment of fewer units. It has seemed to me that for the last three years Africa has been missing the point to which J. R. Williams and L. H. Diehl set themselves with such characteristic energy and carefulness, namely, the utilization of stamps for coarse crushing only, the finer comminution of particles being effected by tube-mill, and it has been apparently left to E. H. Johnson, of the East Rand Proprietary Co., to regain the laurels for the Rand in this direction by his work—referred to later—showing the advantage of increasing their present ratio forthwith from 2 to 10%. Even today the average duty of stamps on the Rand is less than that of the lighter stamps at the Giant in Rhodesia, and at the El Oro in Mexico—and there can be no doubt as to the toughness of the El Oro ore. The El Oro company indeed has been steadily showing the world for some time past how light stamps of under 1250 lb. weight can by the use of tube-mills be made to produce the heavy output above noted. Provided the tube-mills are not

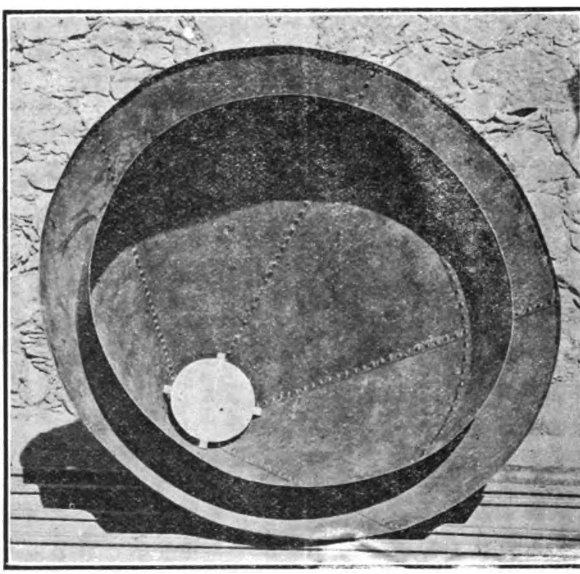
overloaded with pebbles, there seems no question as to the economy in power of such practice, and the better extraction resulting is undoubted. Recently, in a case where it was necessary to obtain the highest possible output with an amount of power severely limited, various schemes were tried, and finally the ratio selected as the most economical for this particular purpose was 6 tube-mills to 36 (1250-lb.) stamps, or 16.6%. This compares with the Waihi Grand Junction practice of 9 tube-mills to 40 stamps, the Santa Gertrudis practice of 10 tube-mills to 60 stamps (1550 lb.), the La Banca practice of 6 tube-mills to 40 stamps (1250 lb.), and the El Oro practice of 14 tube-mills to 100 stamps (varying weights), and altogether surpasses the proportion of 7½% not long ago asserted by a Rand engineer to be the highest ratio in the world. The Rand has been considerably stirred of late by the introduction of the Butters filter. It must be considered a great personal triumph for its introducers. It promises to be even more widely adopted than the Adair-Usher process, and already, as a result of the operation of the plant at the Crown mine, installations are in hand for some other well-known mines of the district.

Rhodesia continues to pursue its own path of progress. First to set the pace in crushing with its 10-ton output at the Giant, it also appears to be the first actually to install and use 2000-lb. stamps, at the Bucks Reef, but its metallurgy appears to be conducted on more enterprising, though possibly more cautious, lines than that of the Rand; thus Pachuca vats are already adopted there, though the caution of the engineers is shown by their care actually to recover all their gold when once they have it in solution, and so two of the most recent plants, the Globe & Phoenix and the Lonely Reef, have installed the possibly to the American mind old-fashioned but safe Dehne presses, of which there is now quite a number of installations in that territory.

Eastern Asia has adopted air-agitation pretty generally, but is not yet altogether enamored of vacuum-filtration. A good general idea of the trend of practice may be gained from a perusal of Mark R. Lamb's recent excellent article in the *Engineering and Mining Journal.*

Concentration.—New methods of crushing have modified former practice, and the professional designer has now to face the problem of concentrating after very coarse crushing by stamps or after very fine sliming in tube-mills. E. Girault was able to deal with the problem characteristically and very neatly at Pachuca. Finding that his impalpably-slimed concentrate would yield to concentration he crushed to a moderate mesh only (10 to 12) and installed Wilfley tables, which he uses as classifiers. He passes all his pulp over them and sends the concentrate to tube-mills, where it is ground with continuous return of the ground material to the Wilfleys until the latter cease to retain the impalpably-slimed particles, which are then in fit condition for cyanidation. But this simple method is unavailable when the concentrate has to be set aside for shipment or other special treatment, and in this case one

has to face the loss in the pulp of comminuted particles which will not be separated out on the vanners. It seems to me that to minimize such loss as would arise from the second grinding of already slimed concentrate particles it is desirable to adopt two-stage concentration: (*a*) tables for a sized product screened from the coarsely crushed (3 to 6 mesh) stamp-battery output and (*b*) vanners for material from the tube-mills. The pulp for (*a*) could be separated out by Callow screens and then sized in cones for concentration as coarse and fine sand. Such a plant indeed may be seen in the Montana-Tonopah, but F. L. Bosqui in his latter plant (Goldfield Consolidated) showed, it seems to me, no little daring and enterprise by boldly doing away with the



CALDECOTT DIAPHRAGM-CONE

tables and relying on vanners after tube-mills only. The decision must have required some courage and Mr. Bosqui certainly deserves credit for the successful working of his scheme. It is naturally to be anticipated that in his new sphere of labor, assisted and not hindered by local talent and experience, he will give no less good account of himself and be responsible for some fresh developments in African practice. Certainly the profession is indebted to him for his Goldfield Consolidated equipment.

While on this subject of concentration, it may be well to add that Walter McDermott—no mean authority on the subject—makes the point that no table can be expected to compare with a belt for slime concentration for the reason principally that the

concentrate once settled should not be disturbed until delivery. In spite of the real success of such tables as Deister No. 3, Mr. McDermott's point must be considered well taken, especially in view of the practice of slime concentration as carried out on tin floors, where ultimate recovery of high-grade concentrate is only possible by the settlement of the heavier particles and the surface washing off of the latter without disturbance of the former; a number of settlings and surface washings being requisite for ultimate success. A well-known authority in Mexico has discovered that his concentration costs him much less per ounce of bullion recovered than his cyanidation, and adds: "Experiments that I have made here lead me to believe that more profitable results can be obtained by thorough concentration and doing away with the cyanide process. You will see that *if* 85% of the metal could be gained by concentration, that method would prove more profitable than cyanide with a 96% recovery." I have italicized the 'if,' as that is just the rub. A small proportion of rich concentrate costs naturally but little per ounce, compared with the expense of cyaniding much lower grade material, but let him attempt to obtain from pulp of the same value (concentrate tailing) by further concentration, the same commercial recovery that he does by concentration, and his figures will alter amazingly. It is becoming more and more the practice for mines to treat their own concentrate on the spot. The cost of shipment plus charges, extras, fines, and deductions, is found to work out at a high figure, and even the Alaska Treadwell, which until recently owned its own smelter, is credited with saving $11 per ton of concentrate by the cyanidation on the spot.

Crushing.—The use of tube-mills has now become so general that there is but little to add to previous remarks on this subject. There has been no startling development in liners, which are mainly of silex blocks or of cast-iron pebble-retaining grooves, with local preferences for longitudinal bars or bolted plates. The reverse-screw in the discharge-end, first suggested by Walter Neal, is becoming more widely adopted as its advantages become known. As stated above, the Rand has been lagging behind Mexico and New Zealand in the number of tube-mills to stamps, but in his latest crushing results, E. H. Johnson, of the East Rand, has been able to obtain a 20-ton daily output per head from 80 stamps (1680 lb.) and 8 tube-mills, using a screen with 8 holes per square inch, with a final product of 90% through 90 (0.006-in.) mesh, of which 50% is treated as slime. Individual tests at Cason and Angelo, East Rand Proprietary, show results of 23 to 25 tons per head with an estimated saving in power of 50%, and it is therefore proposed to equip the 220 stams there with 22 tube-mills. On the other hand, as the *South African Mining Journal* points out (September 10, 1910), the Randfontein Central (Robinson group) is being eqipped with 600 stamps and only 16 tube-mills for the same duty as the 220 stamps with 22 tube-mills mentioned above.

Dewatering and Classifying Pulp.—The wellnigh universal adoption of simple methods for effecting this object is possibly the greatest feature of the progress of the year. Reference has been made in these notes last year[1] to the Dorr classifiers, and now Charles Hoyle, at the Esperanza, uses a classifier which economizes height to an even greater extent than the Dorr and at the same time gives double the output. Mr. Dorr has modified his original gear and has a very cheaply operated and smooth-running machine. John W. Bell, at McGill, has a cone somewhat resembling the Caldecott, but with features of its own. It looks as if there may yet be a new conflict of 'Classifiers *v.* Cones.' At first sight the question is apt to be raised why classifiers, expensive in first cost and using power, should be used instead of cones, which are cheap and operate by gravity, but there is evidently a cogent reason, or the former would not be so extensively adopted. Probably a really continuous and automatic cone, which does not clog or require continuous adjustment, would readily win the day; which only shows that such a cone can not yet have materialized. For dewatering slime, the thickener suggested by Mr. Dorr is being widely adopted. Requiring but little space and little power for greater capacity, 30 or 35 ft. diameter by 10 ft. deep, and under 1 hp. for 100 tons of dry slime per diem, it has a variety of uses, so that it is difficult to predicate where its adoption will end. Used at present mainly for thickening pulp prior to agitation in Pachuca vats (Brown agitators), it is also convenient for use prior to vanners not only delivering a pulp of the required thickness for most effective concentration, but at the same time, without pumping, sending to the vanner the necessary clear water for the operation (recovered from the pulp during thickening). A further use is the recovery of water for sluicing away residue. This is a feature of the new Santa Gertrudis plant. While many thickening appliances have been referred to in recent years (see these notes for last year), no other seems so simple, cheap, and continuous as this.

Agitation.—The Pachuca 'tank' or vat continues on its path of triumph. The Transvaal is the latest country conquered, and in addition to the installation at French Bobs it is understood that the East Rand Proprietary is now putting in a number of them, while various modifications are also receiving much local consideration.

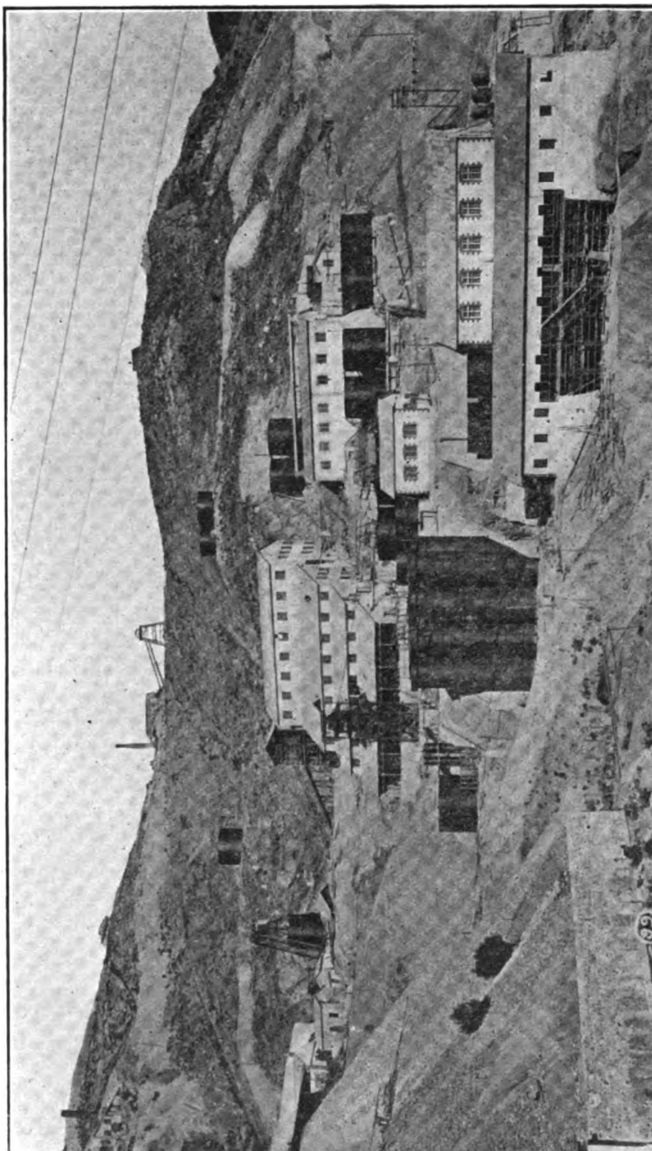
J. M. Nicol, in a criticism of these notes last year,[2] outlined a scheme, then in its infancy, suggested by A. Grothe and M. H. Kuryla (and J. L. Mennell?,) of continuous agitation in series in place of the use of each agitator as a separate unit, and this system has since been adopted at practically every new plant built in Mexico. It saves pumping, loss of level, and indeed is just the scheme for working these agitators in conjunction with the pressure-filters now coming so much into vogue, as the latter can be gravity filled, which makes the operation very cheap indeed.

[1]*Mining and Scientific Press,* January 1, 1910.
[2]*Mining and Scientific Press,* February 26, 1910.

Mr. Nicol's criticism was so valuable, and his encomiums on the process so justified by its universal adoption, that I was content to bless the misunderstanding which evolved the criticism. I criticised a suggested process for treatment by continuous decantation in Pachuca tanks without filtration or any other method of recovering solutions from pulp. Mr. Nicol inferred that I was criticising the new suggestion of continuous agitation, *followed by filtration*, hence his valuable letter. The process I criticised is now as dead as the proverbial doornail; the method Mr. Nicol described will probably extend to the ends of the earth. There were two other criticisms of my notes. E. M. Hamilton[3] wrote to say that he obtained on the mines under his control results superior to those I ascribed to the fixed submerged filter. (Doubtless 'shortly' I shall be able to give the filter a definite legal name instead of risking the result of attaching to it any one of the seven names by which I have heard it described.) This is precisely what one would anticipate from one to whose professional work we are all so deeply indebted. Mr. Hamilton, I feel sure, would never countenance work on such lines as some of that which I came across, and I should like to confirm his remarks by adding that the best work accomplished by any filter of this type which I have yet seen in operation was that which I recently examined at a gravity-filled plant under Mr. Hamilton's professional supervision. There was another criticism, by Ralph Nichols,[4] who is remembered as the engineer that introduced to Kalgoorlie such up-to-date methods as the Moore filter and Merrill precipitation. Mr. Nichols will very possibly agree with me in stating that the possibility of channeling in the Dehne press—which up to date produces a greater proportion of the world's gold output derived from slime than all other filters and methods including decantation, and can therefore be considered a reasonably successful machine— not altogether improbable in a method of filtering by montejus formerly in vogue at Kalgoorlie, is much diminished, if not made entirely impossible, but the more recently introduced system of filling by pumps direct from the agitators and thus giving the pulp no time for that segregation in the montejus which led Herbert Moss to introduce air agitation as a corrective.

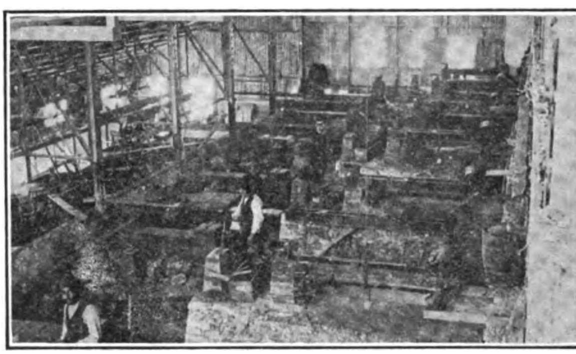
Recovery of Metals in Solution.—A casual observer who has noted the entire absence, of late, in the American technical press of the front-page announcements (regarding the great vacuum-filters) which used to amaze, or amuse, the public, must not thereby hastily assume that the day of vacuum-filters is over. May it not be rather that new processes or new fields are occupying attention formerly too highly localized or specialized? At any rate, Africa is paying a substantial and welcome compliment to the enterprise and business tact of one of the companies. But the fact remains that in the territory in which they have been mostly boomed, vacuum-filters appear to be suffering a setback. Pressure-

[3] *Mining and Scientific Press,* February 26, 1910.
[4] *Mining and Scientific Press,* April 2, 1910.



LA BLANCA MILL, GENERAL VIEW

filters are being widely adopted, particularly in the latest built mills, such as the El Oro, Esperanza, Veta Colorado, Santa Gertrudis, El Tigre, and El Carmen. At Guadalupe the Santa Gertrudis company had a prolonged and careful examination of the various prominent methods, ably carried out by Godfrey Doveton. The method of conducting these tests places the profession under no little obligation to Hugh Rose and the other concerned. The Moore (regular plant already in operation at the mill, the most recent one at that date at Pachuca), the Burt, the Merrill, the Sweetland, and the Kelly (at the outset only), were all represented, and the spirit of good-fellowship among the competitors would alone have made the test noteworthy. In the end Merrill was adopted, with the Sweetland and Burt running very close in order of merit. As Mr. Rose justly remarks, however, conditions at Pachuca must not be held to be truly representative of any other conditions, and "a filter which is most suitable to our ores might



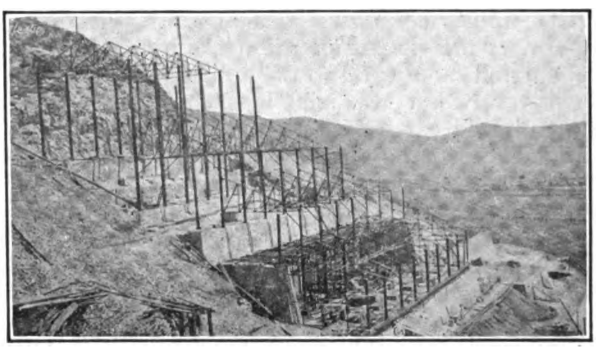
LA BLANCA MILL, CONCENTRATOR FLOOR

be impracticable for others, and vice versa." The result of these tests are so interesting that I append a summary which shows how thorough they were. The total cost per ton of ore filtered, spreading the total cost and royalties, if any, over tonnage to be treated in five years, and including dissolved gold and silver lost, works out as follows:

	Cost, centavos per ton.	Dissolved metal lost centavos.
Merrill	32.6	13
Sweetland	30.7	13
Burt	02.2	5
Moore	88.5	47
Butters	72.1	47

The full report shows the extreme care with which the tests were conducted. Thus, for instance, after determining the screen test, moisture in the feed, time and gauge pressure of cake-making; solution expressed; thickness, moisture, and weight of cakes, with

the rate of flow during caking and washing; volume of wash, and of sluicing water; it was not taken for granted that because the re-washed feed to press assayed, for instance, 83.7 grammes Ag and 0.4 Au and the tailing as discharged 84 Ag and 0.4 Au, that therefore the press had made a recovery of over 99%. On the contrary, the discharged tailing was itself re-washed and the press debited instead of credited with any metal extracted during the process. This amount thus deducted varied from 0.5 to 4, and in one case 6 grammes per ton, or under 0.05 to 0.5% on the high-grade material tested. But even so, the recovery of the pressure-filters was very high indeed; approximating closely to 99%. In justice to the Burt filter, I ought to remark that the figure of costs in these tests, arising mainly from calculations necessarily based on a restricted output in the tests under consideration, must be regarded as quite exceptional and as not applicable to any other conditions. The extremely low loss of dissolved metal in solution is in keeping with the excellent work accomplished by this



LA BLANCA MILL, FOUNDATION AND FRAMEWORK

press at El Oro, where its cost of operation is as low as that of the Merrill at the adjoining mine, the Esperanza. Indeed, one of the most agreeable features of present North American practice is the decision of the owners of the pressure-filters that their presses are not invariably the best for any given ore and that they will permit their presses to be installed only where suitable.

On the other hand, the vacuum-filters have not been idle. Butters can claim the great Dos Estrellas and Palmilla plants, while the Moore has two further installations at Pachuca to its credit, the Purissima and the La Blanca. A feature of the new Butters plant is the large pump and piping used for filling and emptying the filter-tanks so as to reduce the period available for the formation of cracks in the cakes. At the Crown Mines in the Transvaal the Butters engineers claim 0.12 dwt. residue, carrying 30% of moisture for 2d. per ton, which is remarkably good work.

It was a matter of surprise to me, after my experience at Waihi, to find the Mexican traversing filters were just as liable

to the formation of cracks as those of the fixed-submerged type. They seem to manage things better at Waihi, and as the latter type of filter is supposed to require less acid than those used in Mexico, I have looked into the average costs and find that the cloths are cleaned once in every four months at a cost for labor and acid of 0.05c. per ton. This appears to confirm the above view. The Ridgway filters are working merrily along at the Great Boulder and elsewhere, but Ridgway, like every enthusiastic inventor, is busy at yet another filter. He is evidently anxious to have a new model with the washing capacity of the old type and the output of the new. But alterations of types scarcely make for permanency or for popularity. The Arbuckle process, with which persistent experiments have been made on the East Rand, does not yet appear to have been made a success, and it is rumored that the East Rand Proprietary is about to adopt the Butters filter.

Clean-up.—The substitution of Merrill presses for zinc-boxes has become more common this year, and the press has now gained a footing in Africa. At the Esperanza at the time of my visit (May, 1910) the zinc dust consumption averaged about 200 grammes per wet metric ton of ore, or about 3:1 of bullion. The precipitate carried about 60% of bullion and required no acid treatment whatever, but was simply melted down smoothly. The solution precipitated carried about 35 grammes of bullion per ton. It is essential to keep the press absolutely full of solution to prevent oxidation; and Mr. Hoyle, to whom I am indebted for much detailed information regarding improvements in Mexican practice, accomplished this by fitting a gooseneck to the discharge so that this reaches a higher level than the top of the press. In addition to all this, undoubtedly the substitution of a filter-press for all the paraphernalia of zinc-boxes and shavings gives a clean and neat practice, but to one who originally designed and first put into practice the type of zinc-box universally adopted over the world, the old-established practice would appear so firmly based as to not lightly be outplaced. But the scheme of J. S. Colbath, who was in charge of the well-run filter plant at El Oro referred to above, and incidentally the mine and business generally, whereby the use of a Merrill press was supplemented by a zinc-box, appeared to run smoothly to enable a much lower ratio of zinc dust to be used (1:1) with consequent purer precipitate and less trouble in melting down. Mr. Colbath's idea, supplemented by his simple and efficient feeder the design of which he presents to the profession, is not to obtain absolute precipitation in the presses but to rely on a zinc-box to complete his precipitation. It might have been thought that as many zinc-boxes, though possibly with fewer compartments, would be required as before, since the amount of solution is the same, the press merely impoverishing it; but in practice one box seems absolutely affective; indeed, the precipitation appeared to be complete in the first two or three compartments.

Reference has previously been made in these notes to the failure of zinc dust precipitation in Australia, New Zealand, South Africa, and India, but I admit that when face to face with an installation worked by experts, I am much more favorably impressed than with the results obtained by local enthusiasts based on information derived from written descriptions only.

Tilting furnaces are becoming more common. The practice of burning oil fuel in the melting-room without provision for the escape of the products of combustion to the outer air cannot be too strongly condemned. The old furnaces at Kalgoorlie were always connected to a chimney and no inconvenience was noticeable during fusing. The standard furnace runs with a consumption of about 120 cu. ft. of free air per minute at 45-lb. pressure, utilizing two Billow Atomizers Class F.M. and consuming 14½ kg. of Devoes' Brilliant oil per hour.

Among the prominent new mills of the year are the Bantjes (decantation), City Deep (2000-lb. stamps and decantation), French Bobs (Brown agitators, Butters filter, and Merrill precipitation), and Lonely Reef (Brown agitators and Dehne presses) in South Africa; and the Palmilla (Pachuca tanks, Butters filters, and Merrill precipitation), Veta Colorado (Pachuca tanks and Kelly filter), La Blanca (Pachuca tanks, Moore filter, and Merrill precipitation), and Santa Gertrudis (Pachuca tanks, Merrill presses, and Merrill precipitation) in Mexico. The majority of these are not yet at work, although the Bantjes started some months since. In this case some tuning up seemed necessary, but this is natural to all new mills. (See an excellent article by A. W. Warwick, *Engineering and Mining Journal,* October 15, 1910.) The City Deep has not yet started, though the newspapers are crediting the 2000-lb. stamps as being designed for a duty of 9 tons each only. This is probably an error; the engineers responsible may be assumed to realize exactly the potentialities (expressed in these notes last year) of stamps of this weight. It is proverbially difficult to predicate from isolated examples, but it looks as though a high proportion of the new African mills will be designed for filtration instead of for the hitherto impregnable decantation, and that in the new Mexican mills the self-discharging presses will supersede the vacuum-filters in popular favor.

PROGRESS IN CYANIDATION OF GOLD AND SILVER ORES DURING 1911

By ALFRED JAMES

(January 6, 1912)

Introduction.—America still holds pride of place for advanced practice during the year, although the tendency has rather been to settle down and to assimilate the great advances of the past year or two. One is inclined to regard the Santa Gertrudis mill

as absolutely the best example of a silver-gold mill on up-to-date lines. In every way this mill is noteworthy, whether for the trouble taken in the preliminary investigation, the competitive nature of the practical tests, the expert talent assembled from all quarters, or the great success of its working. On the other hand, Africa is coming forward prominently, and there are signs of the more general adoption of methods of great success elsewhere. In this review I last year[1] ventured to suggest that F. L. Bosqui, assisted and not hindered by local talent and experience, would probably give a good account of himself, and the new Brakpan plant, for the success of which W. L. Honnold gives Mr. Bosqui well deserved credit, has signalized itself among recent African plants in being successful from the outset. The Brakpan is notable also as being the first example on the Rand of a mill on modern Mexican lines; thus agitation instead of being by pump transfer is by Pachuca tanks, and the gold in solution is recovered by filtration instead of by decantation, and precipitation is by Merrill zinc dust process instead of by zinc shaving. The first dividend paid in South Africa from a modern mill—Pachuca tanks, Dorr thickeners, and filtration—seems to be that of the Lonely Reef in Rhodesia, to which credit is due for having at work the first plant of this type in that territory or on the Rand. A feature of Rand practice is the apparent difficulty of estimating the value of the gold in slime, pulp, and cyanide solution. That the difficulty does exist is shown by the discussions on the subject at the meetings of the Chemical, Metallurgical & Mining Society of South Africa, and the official report on the Adair-Usher and Butters results at the Crown Mines. At the same time, it seems almost inconceivable that groups of such great responsibility could be misled. Only a year or so ago the Adair-Usher results (readers will probably remember the criticism of the secretary of the company on the remarks in this review) were reported to show a saving of 5d. per ton over the decantation process. The chief mining group was publicly reported, without denial on its part, to have paid a large sum of money for the right to use the process. It now appears from the Crown Mines' official report that the Adair-Usher residue was really 0.423 dwt. instead of 0.225 dwt., the apparent value when samples were taken in the usual way. A fuller account of this interesting report is given under filtration.

In another way Africa has also been prominent. Earlier in the year there were glowing reports on what the Giesecke mill was to do for the industry. Surprise at this was all the greater because it was evident to anyone of experience in ball-mill practice that the mill could not be the success claimed, because it has long been shown that for economical grinding in a tube-mill the elimination of the already slimed material is essential. The Giesecke mill combined a ball-mill with a tube-mill as one unit, the whole of the product of the ball-mill going without separation into the tube-mill portion of the appliance. It was therefore manifestly un-

[1] *Mining and Scientific Press*, January 7, 1911.

economical. In addition, it has been common experience from the time of the original Globe, Lamberton, and other ball-mills that the wear and tear of iron or steel balls, when wet-crushing, was enormous, and there was no reason to anticipate any less wear and tear with the Giesecke mill. Indeed, the Krupp mill is the only one which appears to be able to stand the wear and tear of wet-crushing with iron or steel balls, and even in this mill the wear and tear is very much heavier than when dry-crushing. Last year in this review I ventured to predict, although the City Deep type of mill was then at the zenith of its fame, that a high proportion of the new South African mills would be designed for filtration instead of the hitherto impregnable decantation, and already this prediction seems to be completely fulfilled. It must be difficult to gauge the feelings of such pioneers as the Meyer & Charlton and New Goch, which after putting in the filters which enabled them to show recoveries of 94%, relapsed to decantation—it has been stated for the sake of obtaining a mortar box sample!

India, too, is attacking other methods for the treatment of slime, and the erection of an up-to-date plant is anticipated at the Champion Reef mine; details are not available, but it is understood that the plans provide for Pachuca tanks and vacuum-filtration (Butters type).

West Australia pursues its already evolved practice. Much metallurgical skill has been applied to discover a less unsuccessful system of treatment of the Gwalia Consolidated antimonial ore. On the other hand, H. T. Brett, late of West Australia (of Ivanhoe pan fine-grinding fame), seems to have no doubt as to the correct method of meeting successfully his similar problem at the Globe & Phoenix.

Cornwall, curiously enough, is evolving practice which may later become a potent factor in gold ore treatment. Arthur Richards, who has made no little success of fine grinding of local tin residues with a consequent production of tin only surpassed by the largest mine or mines in the county, has set himself to increase the fine-grinding standard from -200 mesh to -400 mesh, and is endeavoring to find some more suitable appliance than the tube-mill for this particular purpose. Again, Dolcoath has now an equipment of smooth running—giving almost the impression of velvet cushioned—pneumatic stamps which in American or African hands would easily beat the record for tonnage per head per diem. It is a matter for surprise that someone has not taken a battery of these and determined their output on, say, 3-mesh screening with quick drop and most suitable quantity of water, leaving the finer crushing to be accomplished in some secondary appliance such as the tube-mill.

Before proceeding further it is fitting to refer to the satisfactory (from the metallurgical point of view) termination of the East Rand fiasco. It started with a reported huge loss of gold in the cyanide plant altogether incredible to those knowing E. H. Johnson, the able chief metallurgist. As Mr. Johnson was

referred to in this review last year as regaining the laurels for the Rand in the matter of crushing, the accusation or charge was all the more inconceivable to me, and it has given much relief and satisfaction to find that the official report of the committee not only entirely exonerates Mr. Johnson and states there has been no (such) loss of gold in the cyanide plant, but shows how the supposed loss has really been brought about as a result of over-calculation of ore content and tonnage by the mine officials. Mr. Johnson had my sympathy; I now add my congratulations.

New Mills.—Probably the most notable mills of the year are the Santa Gertrudis, Dome, Randfontein Central, City Deep, Brakpan, and Lonely Reef, the last a quite small mill but at the time of its starting probably the most up-to-date in Africa. The Lonely Reef company has already within three months after its starting the mill, declared a dividend. If the City Deep (decantation) be excepted as being really a last year's mill, possibly still somewhat in an unfinished stage, and the Randfontein Central as being of the standard but now obsolescent decantation practice with a vast number of stamps (600) instead of a greater proportion of tube-mills, it will be seen that all these new Mexican, Canadian, Rand, and Rhodesian mills have points in common. They all include stamps of 1250 lbs. or heavier, Dorr type thickeners (Brakpan excepted), air-lift agitation (Pachuca tanks), filtration of slime, and, with the exception of Lonely Reef, Merrill zinc dust precipitation. They appear to have gone to work smoothly from the start (except the Dome, not yet started) and to be giving results which are records for their particular territories. The Stratton's Independence mill, treating dump, and the Alaska Treadwell mill, treating concentrate, are also noteworthy, as will be seen later. The Tonopah Belmont, Modder B, and Palmerejo mills are not included, as details of the starting of these mills are not yet at hand.

Of the Santa Gertrudis mill, it is proper to state that the extraction from the outset has exceeded the estimates, and that it amounts to over 90% of the silver. Exact figures will be given in a paper shortly to be published by Hugh Rose. The mill is of 600 tons capacity and is being brought up to 800 tons by a small additional expenditure. It includes 4 gyratory crushers; sixty 1550-lb. stamps; 14 Dorr classifiers; 6 primary 5 by 16-ft., and 4 secondary 5 by 20-ft. tube-mills; 5 primary 35-ft. and 3 secondary Dorr thickeners; and 18 Pachuca tanks. A feature of the mill scheme is primary and secondary treatment almost all the way through. Even the agitating in continuous agitators is not in one operation. The pulp, when a little more dilute than 1:1, is agitated in a battery of 8 Pachucas, is then elevated by air-lift to a launder, whence after dilution with 4:1 of barren solution, it flows through a second set of Dorr thickeners for a second treatment in 10 Pachucas. Filtration takes place in 4 Merrill presses, the water used for sluicing being recovered by two 35-ft. Dorr thickeners. After classification in 2 sand filters,

the solutions are precipitated by 6 Merrill precipitation-presses in two circuits, one involving complete precipitation, the other and larger reducing the amount of bullion in course of flow. The Merrill slime-press is officially reported to be doing everything expected and to be showing greater capacity than originally claimed and to recvoer a high percentage (98%) of gold and silver in solution at a cost of 2½ to 3d. per ton. This takes no credit for the additional extraction obtained in the press, but actually debits the press with total dissolved metal—under 2 grams of silver—in the residues, although the two grams and more is undoubtedly extra extraction obtained in the press itself during the treatment with barren wash. The above cost does not include royalty or the loss, perhaps 1½d., in dissolved silver and gold. The zinc dust consumption is stated to equal the weight of bullion recovered (1 : 1).

The La Blanca mill, which started at the close of last year, is reported to be obtaining an extraction of 92.8% silver and 95% gold, and to be milling 300 tons per diem at a cost of ₱4.40. The Moore filter is used in this mill and is stated to show dissolved silver to the value of 3 to 4 grams only on the residues from 750-gm. original heads .

Some mystery appears to hang on the Benoni mill. Last year the technical press gave full details of the proposed equipment, the chief new feature of which appeared to be the Way-Arbuckle slime-plant. It is reported that after many days the mill was started in October to comply with certain Government regulations, but that the slime equipment is not yet in working order; doubtless full information will be available later, and it is to be hoped that Mr. Way has made as great a success of the Benoni mill as he previously made of his New Kleinfortein plant; but the latter was based on the treatment methods prevalent at the time, whereas the Benoni slime-plant would appear to be based on practice as yet unproved and untried; except in so far as the unsuccessful experiments at the East Rand may be held to apply to it.

The new Dome mill has had a setback owing to the disastrous forest fire which has necessitated reconstruction. From the published description it would appear to be a mill of standard modern type; forty 1250-lb. stamps, Dorr classifiers and thickeners, 4 tube-mills, continuous Pachuca agitation, Merrill slime-filtration, and zinc dust precipitation.

The cyanide plant at Treadwell is notable as an example of a recent concentrate plant as well as for the extreme attention to detail characterizing the design. W. P. Lass gives a full description of it in the Mining and Scientific Press of October 21, 1911. (Reprinted page 273.) Filtration takes place in Kelly presses and there is Merrill zinc dust precipitation.

The Stratton's plant is left until last, as it is undoubtedly one of the most noteworthy achievements of the year. Readers will remember that the treatment cost of the West Australian telluride

ores was brought down from a very high figure in the early days to as low as 11s. 6d. per ton.[2] It was a brave if not audacious proposal, therefore, to spend a large sum on the erection of a mill to treat telluride dump rock of a grade of 3 dwt. and under! How Philip Argall has succeeded may be seen in the November number of *The Mining Magazine* and also in the account of the November general meeting of the company. He has justified and beaten his estimates, and congratulations are none the less sincere because he has accomplished what few have dared venture. Briefly, Mr. Argall beneficiates the dump at a cost of $1.40 per ton by crushing, concentration, and cyanidation of sand and slime, using gyratory crushers, rolls, Chilean mills, classifiers, concentrators, tube-mills and reconcentration, vacuum-filtration, and zinc-shaving precipitation. It should be noted that the concentrate (3%, containing 44% of total gold in ore) is not treated, but is sold to the smelters on advantageous terms, but the figures include bromo-cyaniding, whenever necessary, of the slime. During the fiscal year 109,800 tons containing 17,288.96 oz. of gold was treated, with a recovery of 7546.8 oz. in the concentrate and 4814.8 oz. from the sand and slime; a total recovery of 12,361.6 oz. or 17.5% for a profit of $65,000.

New Processes.—It is the experience of many that old processes continually reappear under new names with some ingenious change ringing on old chemical or electrolytic methods. It is not, therefore, my practice in this review to refer to newly-proposed methods until the ordeal of practical trial has strengthened rather than effaced them. Usually money and time and trouble may be saved by a knowledge of the result of scientific speculations in the past, and a compilation of these schemes is needed, even though it prove a record mainly of obituary notices. The memory of the old Cassel electro-chlorination process still lingers, and with it the knowledge of the troubles then met in finding anodes that would stand the disintegrating action of passage of current, electrolysis of solution, and wear and tear of agitation of ore-pulp at the same time. It was obvious that the cyanimide process as described by J. C. Clancy must encounter just the difficulties that worried workers with the older process, and from all accounts this trouble has in fact caused delay in bringing the process to a working stage. In the same way the difficulty of the electrolysis of dilute solutions with complete precipitation of metals yet exists, and the troubles that befell Peletan-Clerici and Riecken still await the enthusiastic inventors who chronically rediscover these processes. Long before the advent of the Adair-Usher process it was found that displacement by upward solution was so imperfect as to be unreliable, and that the attempt to recover deposited thickened slime by a screw-conveyor working under water had caused the failure of many a filter or slime-treatment device, and it was, therefore, not a little surprising to learn of the huge

[2]Great Boulder Proprietary—see this review for 1906.

sum (£30,000) stated to have been expended on the Arbuckle experiment at the East Rand. Similarly, the continuous reshuffling of the chemical reaction, solution, or oxidation processes led me to write for the last International Chemical Congress a brief sketch of the results of the employment of various oxides, alkalies, lead salts, lead coating, halogen salts, hypochlorites, accelerators, mercuric cyanide, and alkaline persulphates, with a note of their original introducers into the field of cyanidation. A glance back at history is often sufficient to indicate the part probably to be played by a 'new' method, and that, too, even when the method involves some ingenious application of nice chemical reactions. And as this was my excuse to the reviewer who criticized the omission from 'Cyanide Practice' of any reference to a certain process then advertised but never since heard of, so it may last year and today be taken as my reason for not including in this review of progress in *practice,* comments on ingenious scientific speculations.

Agitation.—The attempt to turn an air-lift agitator into a continuous decanting device have so far had as little success as the old attempt (this year repeated) to filter slime from the bottom of the agitation plant through a filter-bed immediately under the agitator. The feature of the year has undoubtedly been the almost universal adoption of the continuous system of agitation in Pachucas pioneered by A. Grothe, whose able and ingenious article on the principles of agitation in air-lift tanks has given us all food for much reflection. Who before was brave enough to discount what many thought the supreme advantage of the air-lift accelerated agitation, and to argue from the downward travel of the particles only? The article[3] should be read by all. But none the less, the continuous agitation method has brought to light many snags and pitfalls hitherto unsuspected, and from plants on all sides valuable experience is at hand as to means of overcoming the difficulties that crop up. Probably the best course for the intending user is to get into touch with Mr. Grothe at once and to obtain from him the latest results; otherwise he may repeat the experience of a company whose mill is stated in the technical press to have suffered delays of some months on account of the non-working of a locally-devised adaptation of the Pachuca tank. It cannot be too strongly urged that anyone modifying a standard agitator should first experiment with a glass model; he would then find that much of the so-called agitation is merely the escape of bubbles which, cleaving through the volume of liquid, make a great show at the surface, but fail to even slightly modify the descent of all but the finest particles of the slime-pulp.

The originally proposed method of connecting the tanks by inclined pipes is expensive, the more so as by-pass pipes are necessary and the stoppers are awkward to get at, and valves get choked by sand. M. H. Kuryla air-lifts the pulp across any particular tank it is wished to cut out, and Mr. Grothe has central feed and

[3]*Mining and Scientific Press,* July 15, 1911. (Reprinted Page 346).

discharge tanks with the Pachucas grouped around so that any of the tanks can be made No. 1, and thus any particular tank can be cut out without difficulty. All tanks have discharge valves, and thus any tank can readliy be made the last of the series. A simple system of continuous agitation has also been described by Huntington Adams.[4] This method can readily be adapted to existing tanks, and has the advantage of having all connections at the surface of the charge, but to one who has not seen the apparatus in operation the by-passes would appear liable to choke and the system of a variable-cut sampling box at each tank would seem to call for attention and adjustment—otherwise why the adjustments?—whereas it has been shown that the Grothe method does automatically give a pulp of equal screen analysis throughout. The provision, however, in the last tank for agitating a partly filled tank when this is used as a reservoir for filling filters is surely to step in the right direction.

The form of dropping spider shown in the description of the Treadwell plan[5] is worthy of note. A spider which will start up within 15 minutes a densely-packed 30-ton mass of high specific gravity concentrate is an appliance of considerable value in the agitation of concentrate. The Trent tank has come into prominence during the year, mainly by reason of the descrpition of it by J. W. Hutchinson.[6] He describes it as the most serious competitor of the flat-bottom agitator and as a good means of bringing flat-bottomed tanks up to an efficiency comparable to that obtained by Pachucas. The agitator resembles a garden sprinkler, submerged in the pulp, which it agitates by reason of the motion imparted to the radial arms by an issuing stream of circulating pulp. In a personal letter Mr. Hutchinson states the Trent agitator takes 6½ hp. to operate in a charge of 200 dry tons at a dilution of 1½ : 1.

Amalgamation.—In parallel with the practice of concentration ascribed to F. L. Bosqui in this review of last year, whereby the tables immediately after the battery were omitted and concentration limited to vanners or slimers after the tube-mills, several mills in South Africa do away with plates after the mortar-boxes and rely on after tube-mill amalgamation only. It has been foreseen that plates after the mortar-boxes would have to go, because the coarse-crushing now taking place in the mortar-boxes yields fragmentary pulp quite unsuitable for amalgamation on plates, but this removal of the battery plates in no way obviates the difficulty of the effect of cyanide solutions on amalgamated plates, and W. C. Caldecott avoids this by his scheme of dewatering and adding cyanide solution after the tube-mill tables. C. W. Merrill, similarly, at the Dome, is unable, by reason of his employment of amalgamating plates, to avail himself of any advantage arising from crush-

[4]Bull. Amer. Inst. Min. Eng., August 1911.
[5]*Mining and Scientific Press,* October 21, 1911. (Reprinted Page 279).
[6]For description of Goldfield Consolidated Mill, see *Mining and Scientific Press,* May 6, 13, 20, and 27, June 10, 1911, (Reprinted Page 547); also, *Eng. & Min. Jour.,* June 17, 1911.

ing in cyanide solution, and either dewaters his slimed pulp prior to agitation or does not introduce cyanide solution until the pulp is in his press. The Caldecott system yields a beautifully separated and dry (10% moisture?) product adapted to his special requirement—the adding of cyanide solution with as little dilution as possible—but it is questionable whether a valveless table is the most satisfactory or economical method of accomplishing this object. A waste of vacuum at the scraper and a wear and tear of cloths seems inevitable, and possibly a continuous filter of the Askin Nicholas or Oliver type would be more satisfactory in the long run.

A recent advertisement shows that J. V. N. Dorr has fitted a vacuum-filter to the bottom of his well known classifier, presumably to give a dryer product. Mr. Argall, at Stratton's Independence, classifies and dewaters his pulp down to 15 to 20% moisture, treating 200 tons per day with ½ hp. By means of two screen-conveyors he presses the sand and withdraws it with considerable control of fineness of particles and dryness.

For thickening slimp-pulp the Dorr type thickener, and its last variant, the El Tigre, holds the field by reason of its simplicity, its low operation and maintenance cost, and its efficiency. For dewatering slime-pulp the Oliver type filter is probably as convenient as any other apparatus. The method of filtering (dewatering) slime by squeezing it in a folded belt between rollers, described in this review for 1909, is still being experimented with in Australia, and the latest development has been recently described in the *Journal* of the Western Australian Chamber of Mines. A considerable amount of work has recently taken place in South Africa and Australia on systems of dewatering slime by assisted settlement such as that included in the Way-Arbuckle Benoni plant, or that described so fully by Gardner in the May *Journal* of the Western Australian Chamber of Mines. This system depends on the acceleration produced in settlement resulting from the employment of a series of parallel corrugated sheets placed at an angle of, say, 45° to the surface and continuing for some distance (18 in.) below. The corrugations are stated to regulate the flow of descending slime and ascending clarified water without interfering with the settlement of the rest of the pulp. At the Sons of Gwalia an experimental plant is stated to have improved the thickness of the pulp from 26 to 41 per cent.

Clarifying Solutions.—E. J. Sweetland, the Treadwell staff, and C. W. Merrill have all invented presses for clarifying solutions with flowing or sluicing off of the sediment. It is perhaps too early to predicate as to the success of these presses; this must to a considerable extent depend on the nature of the suspended matter in the solution to be clarified. At the Santa Gertrudis recourse is had to sand filters; the well-known Dehne (hand-cleaned) clarifier was specified for the Palmarejo mine, but at the last moment sand filters have been substituted here also.

Crushing.—The race of the Rand to heavier stamps has received a serious check, and the City Deep mill does not appear to be entirely satisfactory. Indeed, the Nissen stamp is reported to have shown up remarkably well against the 'last word' in Rand mills. It may be assumed that for the present Rand practice will for the future assimilate itself more closely to the Mexican and New Zealand practice of increased ratio of tube-mill to stamp. Unfortunately, no figures of performance are available. It was known that the overloaded and over-revolved Rand tube-mills almost immediately after their original installation, passed the margin whereby crushing capacity at the then standard of fineness could be more economically increased by addition of tube-mills rather than of stamps, but on the other hand, the increased fineness resulting from tube-mill grinding has brought about increased extraction and sent up the Rand standard of fineness from 60 mesh to 90 mesh. Indeed, the greatest opponent of tube-mills on the Rand is apparently at last convince and has not only laid down a record ratio of tube-mills to stamps for the Rand at the Benoni, but it looks as if the New Kleinfontein plant itself was following suit on the same lines. We may thus take it that, in spite of the greater horse-power used in grinding through tube-mills, the practice is one that pays.

In this review reference has been previously made to the low ratio, 2 to 4% of tube-mills to stamps prevailing on the Rand as against 16.6 and even 20% elsewhere. An instructive table is given in the *South African Mining Journal* for September 2. It shows that the mills with a tube-mill to stamp ratio of about 1 : 18.5 crush from 11 to 20 tons per head per diem; when the ratio is 1 : 20 the output is from 10 to 13 tons per diem, with a ratio of 1 : 25 the output descends to from 9 to 10½ tons per diem, and those with only 1 tube-mill to 30 or more stamps show only 8 to 9½ tons per diem; but the value of the table is diminished by absence of any note of the weight of the stamps or of the total horse-power employed.

The trial of the Nissen stamp at the City Deep has provided some extraordinary results; it would scarcely be too much to ascribe a superiority of over 30% in crushing capacity, weight for weight, for less horse-power and repairs, and that, too, taking a considerably coarser feed.[7] It is evident from the above results that a limit will be put to the development of heavy stamps in batteries of 5 heads per mortar-box, as the heavier the head the greater should be the area of shoe and die. A noteworthy point is made by J. W. Hutchinson,[8] who states that instead of increasing the Goldfield Consolidated mill by 40 stamps and 3 tube-mills as originally proposed, they obtained the desired output by the addition of 6 Chilean mills only between the existing stamps and tubes. Battery-screens of 4 mesh were substituted for the existing 12-mesh with a resulting duty of 8½ tons per head. This scheme

[7] *South African Min. Jour.*, Nov. 4, 1911.
[8] *Mining and Scientific Press, loc. cit.* (Reprinted Page 548).

saved a month's time, $100,000 in money, and yet increased the capacity of the plant by 40 per cent.

Fine Grinding—E. J. Ball, in a paper before the Institution of Mining and Metallurgy, shows that present African practice is to grind 130 tons of product passing 90-mesh screen for 75 to 90 hp. with a pebble load of 8 to 14 tons, depending on the wear of the liner. This gives a factor for comparison with the data of El Oro, Waihi, and other fields previously given in these reviews. A standard mill in South Africa consumes 7 hp. per ton of pebbles and produces 10 tons of ground sand per ton of pebble charge per diem, or, roughly, 1½ tons per horse-power through 90 mesh. El Oro produces 1.2 tons per horse-power through 200 mesh. Waihi figures are greater than these, but include the slimed ore from the mill. The best feed appears to be 30 tons per ton of pebble load at a moisture of 38.8% (Walter Neal). The pebble load is shown by Mr. Ball to be the best when the mill is filled to just below the centre of rotation, while the speed of rotation should be 10% greater than that given by Davidsen's formula. Now Balls tests were so successful that he brought the efficiency of a makeshift tube-mill—an old chlorination barrel—from 3.87 (S.8) to 8.82 (V) and gave results comparable with those from present-day specially-designed mills. It is to be hoped ere long similar tests may be made on the economical limit of coarseness of feed and of irregularity of mine rock (substituted for pebbles).

There is but little fresh to chronicle in tube-mill practice. As previously suggested, Walter Neal's reverse screw outlet is becoming more and more widely used, and bar liners are displacing all other forms. The ideal liner would seem to be an El Oro liner with replaceable ribs or bars, so that the backing need not be wasted when relining is necessary. Richards (Cornwall) points out the necessity of having small pebbles as well as large in the charge of tube-mills newly started. After continued running, the necessary proportion of small pebbles is mechanically produced, but at the outset he finds output poor unless the charge is made up as stated above. Allen's point of rounding off mine rock prior to its introduction into the mill appears well taken, and is supported by the work of Davidsen and the experience of Neal. The original ten machines at the Great Boulder have treated a million tons of slime to date—a pretty good record. The latest type of machine, however, has adjustable periods of revolution enabling the cycle to be altered at will or even with the age (and lessening filtering capacity) of the cloths. It works automatically, the adjustment being by water-balance, with feed varied at desire, and is showing a good capacity even with the difficult St. George kaolin. On Great Boulder pulp the latest Ridgway type handles daily over 250 tons per unit.

Filtration.—The fact that all the new great mills of the world referred to under 'New Mills' treat their slime by filtration, evidences how absolutely this method has displaced decantation.

From West Scotland Street to Karangahake, from Karangahake to Kalgoorlie, from Kalgoorlie to the uttermost parts of the earth, filtration has progressed on its triumphant march; except for the Rand, where the New Consolidated Langlaagte mill is reported to be designed for decantation, it may be regarded as universal. It is noteworthy that all the typical new mill plants noted, except the Brakpan, have pressure-filters in spite of the great advances made in vacuum-filters, mainly on the lines indicated by Crush in 1909, namely, the adoption of gravity-filled huge pipe-line plants emptying or filling in three minutes and thus avoiding or lessening cracks, in place of the centrifugal-pump-throttled slow-filling systems described in the technical press of that date.

Pressure-Filters.—The Dehne still easily ranks first for number of installations, and is the press adopted by such new mills of the year as the Lonely Reef and the Matabele Queen. The Merrill has given a remarkably good account of itself at the Santa Gertrudis, and the Kelly has been adopted for the treatment of concentrate at the Goldfield Consolidated and the Alaska Treadwell new mills.

Vacuum-Filters.—A spell of holy calm has supervened after the enthusiasm of past years. The Moore is reported to have been adopted for a further plant at Pachuca by an old friend there. The submerged filter has undoubtedly advanced since the adoption of the principle of rapid emptying and filling. Charles Butters has been energetically moving in new territory, Africa and India, and the official report of the Crown Reef plant may be regarded as a veritable triumph. Years ago the Denny brothers pointed out the loss in dissolved gold in the decantation residues and their accompanying solution. It was reported that the total maximum further possible saving was only 8d. It seemed impossible to break down this stand and to save gold for people who thus plainly expressed their belief it was not there to save. Even the Ridgway demonstration which reduced decantation residues from 0.5 and 0.6 dwt. to 0.12 dwt. failed to shake this attitude, and when the same demonstration showed indubitably that gold was still being dissolved after the pulp had been agitated and treated by decantation, even then the possibility of this was locally denied. The official report on the working of the Butters plant at the Crown Reef should put an end to all this. The true average of Adair-Usher residue is shown to be 0.423 dwt. instead of the 0.225 dwt. stated. The Butters plant recovers the dissolved gold, and also by the use of barren solution wash further gold, to the extent of 0.332 dwt., 1s. 5d. per ton, and may under normal conditions be reasonably expected to show a gross saving over decantation of 0.20 dwt. (10d.), or a net saving of 0.12 (6d.) per ton of slime filtered with a treatment cost of 9d. per ton filtered. It may be reasonably anticipated that under normal conditions the Butters cost will not amount to anything like this figure. Under demonstration conditions extra cost will have naturally been incurred to bring the figure above normal. It

will be remembered that the fixed submerged costs were given as from 6d. to 10d. per ton in this review for 1909, and with the huge-main gravity-filling system now being adopted at all plants, it may be expected that costs will be still further reduced.

The following cycle is given for comparison with the old ones published in 'Recent Cyanide Practice.' It well shows the advantage of the gravity-filling huge pipe-line plant to which an old pump-filling plant was converted, and is only possible, of course, by the use of the hydraulically-operated valves first included in the rapid-filling Crush installations. By the use of 20-in. pipe-lines the cycle is completed in one hour, as follows, the time being stated in minutes: Fill, 3; form cake, 19; discharge pulp, 3; fill, wash, discharge with barren solution, 21; water-wash and cake discharge, 11; total, 57. This may be compared with a cycle of 2 hr. 5 min. by the old pump-throttling system. By the new method, not only a better washed product is obtained by avoiding the formation of cracks by long exposure of the uncovered cake, but also the unparallel output of 7 dry metric tons per leaf per 24 hr. is achieved. The industry is indebted to C. W. Van Law, of Deal del Monte, not only for the record established for the above work, but for much serious original investigation into the question of the continuous solubility of gold and silver in pulp after agitation, and even after filtration. Mr. Van Law establishes the necessity for dilution with barren solution after agitation, and for slight re-agitation to obtain best results. An extract from a letter dealing with one of the largest and most recent submerged-filter plants of the day gives point to the remarks above regarding the evils of throttling and the advantage of the Van Law cycle. "The bad feature of it is that the excessive time of change from slime to wash is worse than wasted, for it allows the cakes to crack. When giving a water-wash the time lost in manipulation, exclusive of discharging the cakes, is 36 minutes on each cycle. Allowing an hour and a half per cycle, that shows a 20% loss of capacity as compared with changing in 6 minutes."

J. W. Hutchinson's articles on the mill at Goldfield[9] have already been the cause of the saving of many thousands of dollars by submerged-filter operators. He most carefully prevents unavoidable loss of the water-wash into which his cakes are discharged, pumps off all possible water to a storage tank, and uses it for diluting his Pachuca charges. Any enrichment of the solution is thus retained in the mill-flow for ultimate precipitation. His slime residue flows to the dam as a very thick pulp, carrying little more than 33% moisture. Whether this method is an improvement on that of A. H. Smith, also at Goldfield, described in this review of 1909, is perhaps a moot point. Mr. Smith runs his residues into a pond with wash solution, settles, and decants, and to this extent may recover a larger proportion of value contained in the cake than Mr. Hutchinson, who decants from his tank; on the

[9] *Mining and Scientific Press, loc. cit.* (Reprinted Page 547).

other hand, Mr. Hutchinson's method should be less costly and less liable to loss. But it would look as though he could make a higher recovery by following the example of Mr. Smith and decanting after the cake has been agitated with wash instead of after a dropping only.

The Ridgway has not been entirely resting on its laurels. From New South Wales comes the account of its having ousted a grouped-frame filter of well known type, and a plant has also gone into British Columbia. But in Africa its sponsors, having demonstrated its efficiency, have preferred to wait for a coming to terms by the mining groups rather than to undertake further expense for a problematical minute remuneration. The Thompson-Davis-Crush plants have this year introduced a further refinement by a contrivance which avoids the use of air (with its inconvenience of frothing) for agitation. Drage has brought out a filter in Australia which somewhat resembles a Ridgway set vertically. The Way-Arbuckle combination appears to have been long endeavoring to patent a somewhat similar appliance with double screen-area, judging from U. S. Patent Specifications issued June 6, 1911.

Heating Solution.—Year by year I refer to this in the hope that it may lead to a general communication of views as to the benefits or disadvantages of this course. The difficulty of precipitating gold from solution on the Rand in the winter months emphasizes the necessity for some such exchange of experience. The method suggested by Swen of reheating the air employed in air-lift agitators seems a simple and feasible plan of warming solutions and at the same time making less air necessary for a given duty. C. T. Rice[10] refers to an increased recovery of 2% at the Desert mill resulting from heating the solution.

Precipitation.—The Merrill zinc dust method has been gaining ground as a result of the careful technical supervision of the inventor and his associates. The method has been still further improved by the adoption of a two-circuit plan, the main circuit for reducing solution content during flow, and the secondary circuit for the complete precipitation of solution to be used as wash, etc. By the adoption of this system the consumption of zinc dust has been reduced to 1 :1 of bullion; indeed, even a lower consumption is claimed at some plants. The system has also shown to advantage in the precipitation of cupriferous solutions. The continuous feed of zinc dust acting at once, instead of becoming more sluggish after a while, as with copper-coated zinc shaving, has given remarkably good results in dealing with such solutions, as witness the Goldfield Consolidated plant when the precipitate contains 43% copper. Undoubtedly, a drawback to the system is the large amount of oxide and impurities, perhaps 15%, added to the precipitate, all of which has to be fluxed off unless resort is made to acid treatment. Another drawback is undoubtedly the clogging up of the pipe system between the feed and the press with bullion

[10] *Eng. & Min. Jour.*, June 17, 1911.

precipitate—this latter difficulty should be easily overcome, and I am expecting Mr. Merrill to make his hold on the process greater than ever by supplying a specially prepared pure zinc dust.

General.—Silver mills seem to have bulked more largely this year in the matter of advanced practice than gold mills; silver sulphide is not so easily attacked by cyanide solution as metallic gold, and this fact probably accounts for the advanced position in practice held by Mexico and for the extraordinarily high efficiency in gold recovery of the mills there. The keen enthusiastic emulation of the Santa Gertrudis, Real del Monte, San Raphael, La Blanca, El Oro, Esperanza, and Dos Estrellas mills reminds one of the similar position in Kalgoorlie some years back, and cannot but make for great technical progress.

SLIME FILTERS IN NEW SOUTH WALES

(April 6, 1912)

The Editor:

Sir—In Alfred James' able 'Review of the Cyanide Progress for 1911,' he states that the Ridgway slime filter has ousted a "grouped frame filter of well known type in New South Wales." He is certainly misinformed. There are only two vacuum slime-filter plants at work in New South Wales at the present time. One is at the Occidental mine, Cobar, being the first Moore filter plant erected in Australia, and still working most successfully; and one No. 2 type Ridgway is working on some residues at the Mitchell's Creek mine, Wellington. The only other vacuum filter plant that was erected in this state was a Wilfley drum, at the Myall's and Peak Hill mine, but was found unsuitable. The largest gold mine in this state, the Mount Boppy, is now erecting a new slime plant and adopting the Moore filter.

A. J. CHAPPLE.

Parramatta, New South Wales, February 15.

PROGRESS IN GOLD-SILVER ORE TREATMENT DURING 1912

By ALFRED JAMES

(January 4, 1913)

Costs

It is singular that for the present year, as in 1906, fine sliming and the treatment of slime have still been the main centres of interest. It is difficult to believe that details of practice determined so long since, and even then capable of producing records not since materially modified, remain of chief interest. Readers of this review may have noticed that since 1906 costs have no longer been a feature in it; this is for the simple reason that they have not yet

notably bettered, at any rate at the mines then referred to. Thus in 1906 rock-breaking was stated to amount to 1.71d. per ton at the Lake View and 1.87d. at the Ivanhoe. At the latter mine the figure for the past year has been 3.93d. Milling at the Ivanhoe was 1s. 9d. per ton; it is now 1s. 8d. Roasting was 2s. 4d. per ton at the Great Boulder; it is now considerably higher. It would be interesting to know if the so-called improved furnaces are really improvements and, if so, why they cannot beat the Great Boulder 1906 record of 2s. 4d. per to non roasted sulpho-telluride ore. Cyaniding by agitation was then 1s. 4d. per ton at the South Kalgurli, including 7d. per ton for KCN; has this cost for agitation treatment been since bettered locally? Dehne filter-pressing at 1s. 6d. has been undoubtedly cheapened, both in Africa and Australia, to 10d. and 1s., but the Ridgway filter, at that time stated to be giving high recoveries for 4d. per ton, is still preëminent with its total cost of 1d. per ton on a million tons of low-grade pulp and 2d. per ton on high-grade pulp (value say £3 to £4 sterling) not usually treated direct in filters of the submerged vacuum type. It would be as well to again collate costs; old figures become stale, and I shall welcome notes of actual cost per ton handled for crushing, milling, tube-milling, cyanidation of sand, cyanidation of slime ,agitating, vacuum-filtration, and filter-pressing. It is a solace to think, however, that in spite of increased costs, the percolation of 1 dwt. sand is still carried on at a profit.

General Tendencies

The reference above to vacuum-filters reminds me of the possible temerity in referring by name to the filter of the best known exponents of the fixed submerged type, but decisions of local courts do not count with the industry to the same extent as the enthusiasm, energy, and skill displayed in forcing this submerged filter on the market. Long ago it was hoped that the pioneer work of G. P. Moore, and the irrepressible enthusiasm and skilled energy of Charles Butters, might result in a combination that would benefit the industry the whole world over, and yet leave room for a liberal reward for the combination. What the courts may or may not decide does not modify the feeling of which that hope was the expression; and if the combination went further and included Askin Nicholas, C. W. Thompson, H. P. Barry, G. Ridgway, and J. V. N. Dorr (surely all pioneers in their way) it would be still more solid.

A feature of the year has been the reported closing down of the last surviving chlorination plants. The expression at the World's Columbia Exposition twenty years since, "Cyanide is king," has surely been amply justified. A practice probably doomed to speedy disappearance is that in Africa of having separate sets of vats for the collecting and possible preliminary leaching of the sand prior to the main treatment. If the new Rand mills, such as the Geduld, be considered, it will be found that reliance is placed on a single set of vats, and this surely is justified by the results of

the Princess and East Rand plants. At the latter E. H. Johnson shows some of the highest extractions on the Rand.

Decantation, too, is dead as a process to be adopted at new mills. These are all designed for filtration. Indeed, it is only too evident that Africa is awakening from the benumbing mechanical influence that, up to recently, seemingly confined its metallurgical aspirations to the better preparation of the sand for leaching, and the feeling of satisfaction with local standards is giving way to the keen desire to know more of the successful work of American, Mexican, and Australian confrères in the solution and recovery of the precious metals, and to adopt such methods locally. Thus the Rand has now in active hearness its F. L. Bosqui in addition to its W. A. Caldecott and E. H. Johnson, W. R. Dowling and M. Torrente; Rhodesia, which has ever been on the alert, has brought over A. W. Allen from Mexico in addition to its many celebrated Australian specialists, and even recently sent one of its Australian experts to Cripple Creek to decide some point of advanced practice in which it wished to indulge. This is surely as it should be. The day has gone when men of note can go to New Zealand, Australia, Mexico, and the United States, and come back only to decry what they have seen. Surely the better the expert the greater his capacity for recognizing what is good in outside practice.

Generally, progress for the year has been along developed lines, with no brilliant surprises. Possibly the work of our friend Charles Butters at the Nipissing (to be referred to under 'Amalgamation) is the most daring technical exploit of the year.

Changes on the Rand

On the other hand, owing to the attractiveness of the newer practice of tube-milling to –200-mesh, simple clean classification, air-agitation, slime-filtration, and zinc-dust precipitation, it has been easy to lose sight of steady progress on older lines, such as is evidenced in the design and construction of the mill of the Roodepoort United Main Reef G. M. Co. At a period when new mills on the Rand were under a cloud (relieved later, it is true, by the starting up of the Brakpan and Modder B), the Roodepoort mill demonstrated how heavy stamps and high duty could be combined with economy in design, in power, and in working costs. Even the huge and carefully planned Randfontein mill, with its cleverly designed scheme for handling ore, must yield in attractiveness to the Albu mill, in which may be noted the soundness of the judgment which has enabled S. H. Farrar, F. Bulkeley, and M. Torrente to avoid extravagances noticeable elsewhere and yet to include just those things (big steel bins, almost vibrationless heavy stamps, tube-mills of large diameter and specially determined length with simple linings, and tailing-wheels) which make it one of the most cheaply operated mills on the Rand. Why should local engineers put in power-eating centrifugal pumps with most costly monthly renewals (£60 per month or more), with uneven delivery of pulp to

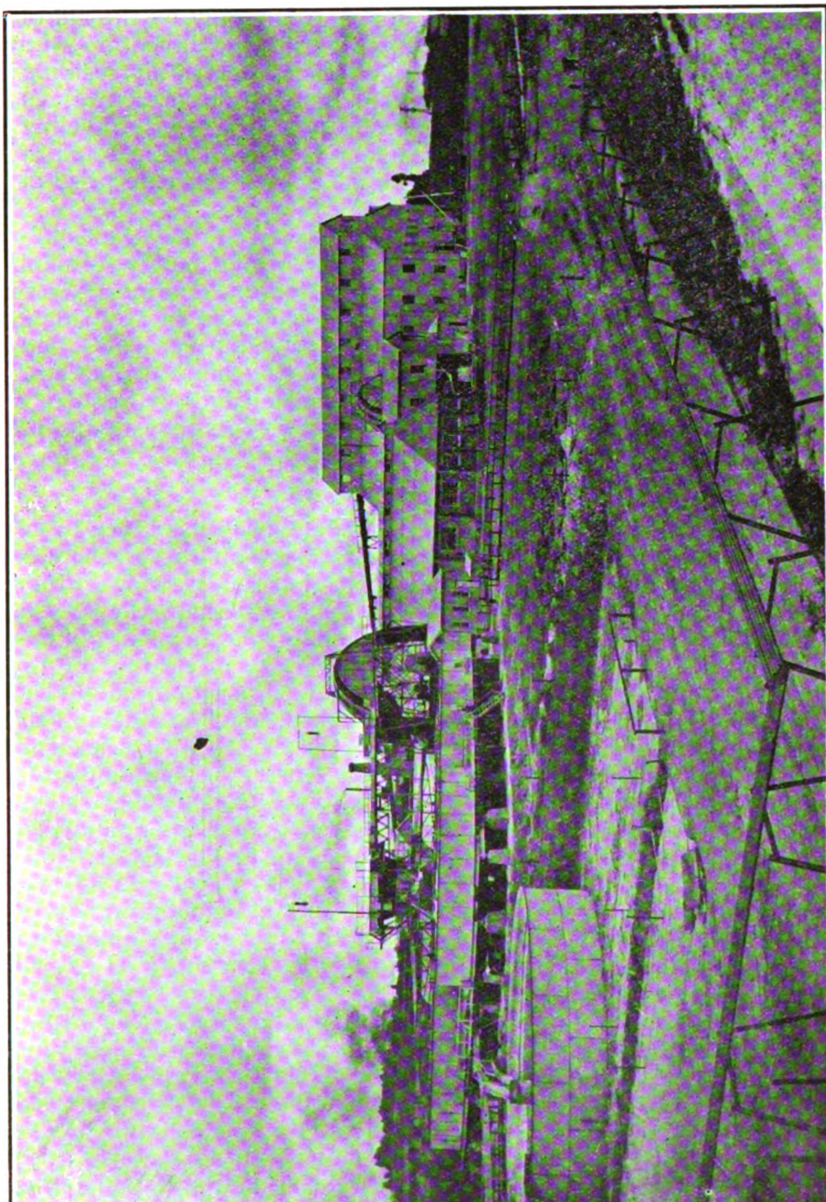
the spitzkasten, and with an awkward tendency to go wrong with coarsely crushed pulp, when tailing-wheels at less than £10 per month for repairs do all the work with a steady flowing discharge on even the coarsest crushed material?

The Roodepoort mill crushes 20 tons per head per day, and does this readily with only one feeder per battery. This mill is stated to operate for 14 kw-hr. per ton for everything from breaker stations to clean-up and blacksmith-shop, as against the 23 to 30 kw-hr. stated to be usual in African practice. In other words, it crushes and beneficiates 1000 tons of ore per day with about 700 hp., and a peak load under 1000 hp. The mill consists of 100 stamps of 1900 lb., only 50 working. The tube-mills are 6 ft. in diameter and 16 ft. long. Stamping costs 11d. per ton. Milling (complete), including sorting, crushing, stamping, tube-milling, amalgamating, and retorting (everything up to the cyanide plant) costs 1s. 6d. per ton.

The group (Albu) responsible for this mill is also responsible for many of the improvements in Rand practice, including the Denny Bros.' abandonment of long (30-day) percolation of coarse sand particles, the introduction of tube-mills, and of the filtration of slime; and it has thus demonstrated that keenness to investigate and adopt new methods (proved elsewhere) may go hand in hand with economy in first cost and in working expense coupled with large outputs and low power bills. Undoubtedly, up to the present the Roodepoort mill may be regarded as the best example of a modern mill on the Rand of the type general to the field; that is, without air-agitation or slime-filtration.

But, as stated above, this type of mill is now moribund. S. H. Pearce, at the latest Crown Reef mill, takes the precaution of a final air-agitation for his slime prior to filtration in the Butters filter, which has produced so great a change in Rand methods. E. H. Jonnson, at the East Rand, was one of the first to approve of filtration and the first to adopt continuous agitation in Pachucas. But the 'Doctor' of the Rand must be F. L. Bosqui; one almost fears to state how low the Bantjes recovery was before he took it in hand, or how great an improvement he has effected in City & Suburban results. African 95% total recoveries have always compared most favorably with those obtained elsewhere. Mr. Bosqui has been evening up the laggards, in one case by recovering increased gold from the slime, and in another case by adopting more modern methods for the treatment of the coarse classified sand.

While writing of Africa, I cannot but wonder at the paucity of practical result of the Mines Trials' Committee's work. Local inventors have from time to time declaimed its methods, but the industry may well ask the reason for there having been so little fruit for so much effort; so little result in an industry obviously decadent in solution and recovery methods, until the arrival of fresh blood from America infused fresh energy. The Mines Trials' Committee recommends relatively large payments to local promoters of tube-mill cones, nozzles, feed, or lining ideas; but what



ROODEPOORT UNITED MAIN REEF G. M. CO. MILL

has it done to encourage foreign inventors to bring in the fresh ideas so much needed in the industry The Nissen stamp has been forced to a successful issue by an outside group, only by the aid of the unwearying and continuous attention of the inventor, who went to the expense of jeopardizing his other interests by residing on the Rand. But the amount of business up to the present given by the Rand to the inventor will probably not pay his traveling expenses, and if ever the Nissen stamp is adopted by the Rand at its mills yet to be designed, it will be because of the enterprise of Rhodesia in widely adopting it.

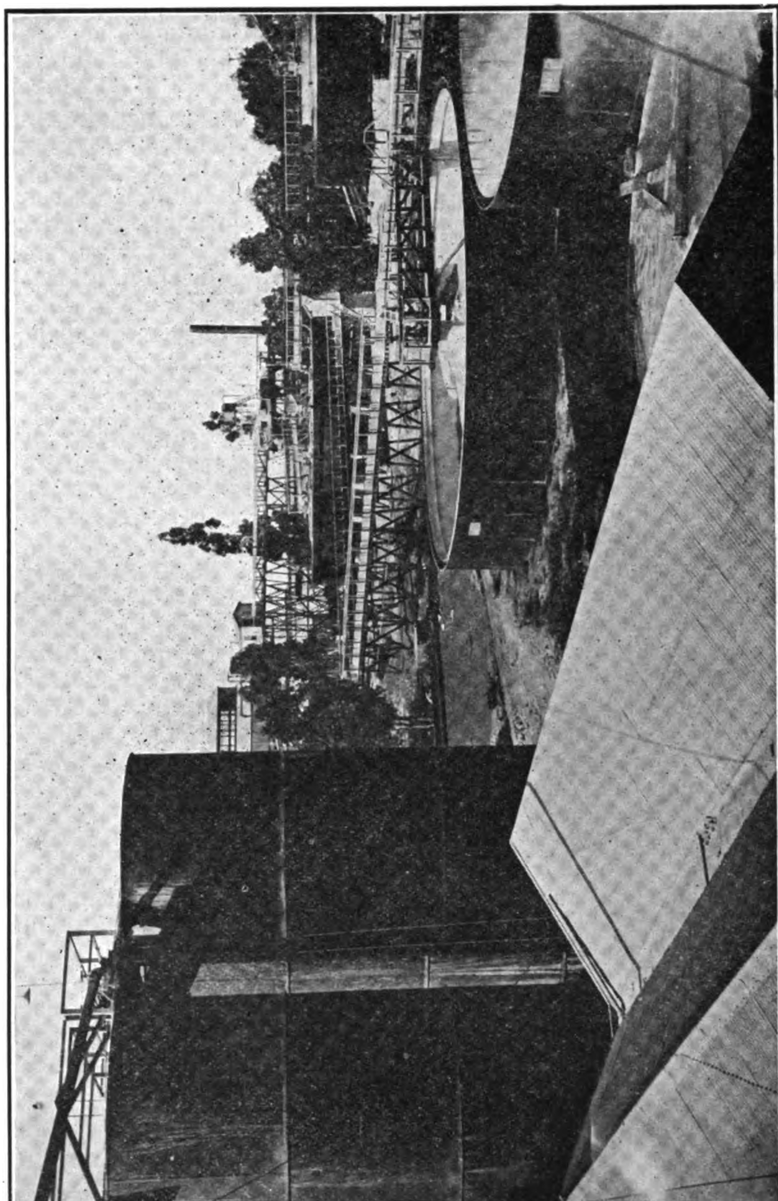
Whatever the standpoint of the industry, it may be confidently asserted that any non-publication of results on the ground of communicating useful ideas to non-contributors is a most suicidary policy. Have the contributors really benefited in any single point? The fact is that the secrecy of the committee avoids that effective criticism which alone can lead to the highest and best results. Let the committee either be disbanded or reorganized, with power to immediately publish the results of its investigations.

Other Countries

From Africa to India is a far cry. Some delay has taken place in the introduction of the newer methods (tube-milling and slime-filtration) referred to last year, but come they surely will. From New Zealand comes the story of a tailing plant, perhaps the largest in the world, ably described by E. A. Banks,[1] who incidentally explains how the injurious carbonaceous content is cleared off by vanners. If only those undertaking similar work would be guided by the successful results so frankly described, much avoidable loss would be saved. Thus a similar project in South America has twice invited and experienced failure through relying on pumps for dredging of the submerged tailing. A. W. Allen's article[2] shows how the greatest success has been achieved by using an air-lift in place of the pumps. The antimonial problem at the Gwalia Consolidated, in Australia, does not yet appear to have been solved, although it has been before many of the leading metallurgists of the day. The latest idea is apparently to work on lines similar to A. Richards' tin volatilization process.

From Mexico there is but little to chronicle this year. The disturbed internal condition of the country is naturally responsible for a setback, but the Santa Gertrudis has signalized the first year with its new plant by paying a good dividend. It has also increased the capacity of its mill to 850 tons per day, which is a good criterion of the success of its methods, referred to in detail in the review of last year. From the United States there is word of new mills containing much of interest, even though there may be no departure from the methods described last year. The Colburn mill, which was watched with especial interest, has apparently

[1] *The Mining Magazine,* August 1912, p. 121.
[2] *Ibid,* April 1912, p. 284.



MODDER B CYANIDE PLANT

had a setback. Good news is at hand of A. L. Blomfield's work at the Golden Cycle mill; it is hoped soon that the technical press may contain an account of the interesting sand-feeder used in the cyanide vats. Phillip Argall has been continuing his very successful work at Stratton's Independence, beneficiating refractory low-grade telluride ore at the remarkably low cost of $1.50 per ton. A. J. Clark and W. J. Sharwood's description of the metallurgy of the Homestake[3] is a valuable contribution. Attention is specially invited to the notes by Whitman Symmes[4] on the Mexican mill, Virginia City. The criticism of equipment and methods of a mill employing such up-to-date apparatus as Trent agitators and Oliver filters and zinc dust precipitation is valuable and corroborative of impressions gained elsewhere. At Tonopah important improvements have been made and the Tonopah-Belmont mill is widely praised, though no detailed account of its work has been published.

Details of Practice

New Mills.—As stated above, there have not been many new mills designed this year; perhaps Rhodesia has been the most active centre. There the Shamva mill uses 56 Nissen stamps, 9 tube-mills, and the Butters filter (336 leaves). The Falcon uses 36 Nissen stamps. The Cam & Motor includes roasting furnaces, Argall driers, and vacuum-filters. The Antelope also has Edwards furnaces and Dehne presses, with ball-mills and pans. The Bell has roasting furnaces, ball-mills, tube-mills, and Dehne presses. The tendency is wellnigh universal for new mills on the Rand to have 1900-lb. stamps, 10% tube-mills of size settled by M. Torrente (that is, wider and shorter than the Rand mines pattern), Brown agitators, and Butters filters. The practice is gaining ground of direct fall from stamps to tube-mills, thus obviating the present custom of elevating the very coarsely crushed pulp so destructive to pump linings. The new Van Ryn Deep mill is to have the usual gravitation stamps and the Butters filter. In Mexico, the Pozos (Augustias) have Pachuca vats and probably vacuum-filters (Moore type); Paterson agitators and Oliver filters are being also adopted. The Palmarejo new mill has been delayed by the revolution, but as no description of this has yet appeared, and as certain features are still novel (notably the sectionalization of stamps so heavy as 1250 lb. for mule-back transport), some reference seems not out of place. Limited by the water-power available (only 600 hp.) and by the difficulty of transport, advantage was ultimately taken of the most approved methods. The employment of such heavy stamps was found possible without the necessity for light cam-shafts by limiting the length of the latter and having three stamps only to a mortar-box. Samples are automatically taken between the feeders and the mortar-boxes and conveyed by belt to the sampling-floor. As originally designed, concentration took

[3] *Bull.* Inst. Min. & Met., Nov. 1912.
[4] *Eng. & Min. Jour.*, Oct. 12, 1912, p. 701.

place both before and after the tube-mills, but it was found possible to do away with the coarse sand tables and to rely entirely on the slimers. Dorr classifiers were originally specified for the tube-mill feed, but were ultimately displaced by Bell cones with sight regulation. The percentage of tube-mills (16 ft. by 4 ft. 4 in.) to stamps is 16.6. The tube-mills are sectionalized by the system of longitudinal channels originally tried with so much satisfaction at Redjang Lebong, in Sumatra. Neal's discharge is provided. A conveyor keeps the tube-mill floor supplied with pebbles or ore. After concentration the thickened slime pulp flows to six standard Pachucas, followed by two storage decanters. Filtration is by Merrill press, with clarifiers for decanted and other solutions. Precipitation is by Merrill press in two circuits with provision for a final flow through zinc shaving, as at El Rayo. Precipitate is melted in revolving tilting furnaces of the Kalgoorlie type. In Australia the new mills, the Bullfinch, the Corinthian North, and the Ora Banda, are all on filtration (Ridgway) lines.

Agitation.—S. D. McMiken has shown[5] how at the Komata Reefs he is able to run ten Pachucas, 38 by 10 ft., with 8 hp. Modifications in internal air-lift pipe lengths failed to improve the extraction. In South Africa the Pachuca has been adopted at some of the best known mines, notably the Robinson, Crown, Brakpan, Modder B, and East Rand. Demonstration runs have also been made by the Trent and Hendryx agitators. The published results of the latter show high recoveries in a very short time, with a very low consumption of cyanide. It is to be wished that W. E. Hendryx had carried his demonstrations further, so as to show precisely what advantage the industry would obtain were it to agitate the total product in an agitator in place of percolating the sand as at present.

Generally, experience in gold-ore treatment, as contrasted with silver-ore treatment, indicates that the fine grinding of sand and the agitation of the finely ground product do not yield enough higher results over ordinary percolation to justify the extra cost of equipment and treatment as slime product. Therefore the gold mines find it pays best to still separate the sand and percolate it. Mr. Hendryx' results show a residue of 0.05 dwt. from agitated slime as against a residue of 0.281, 0.35, 0.25, and 0.55 dwt. for sand and slime mixed. If it be assumed that half the mixed residues are slime at 0.05 only, it brings the sand residue to a very high figure.

The collapse of the Way-Arbuckle agitating plant has been so fully referred to[6] as to need no further comment, except as to the extraordinary condition of affairs which could permit the occurrence of a foreseen and foretold disaster of such magnitude; the loss has been reported to have been as much as £200,000. A contribution to the technology of the subject is that of J. E. Alley.[7]

[5]*Mexican Min. Jour.*, Nov. 1912, p. 19.
[6]*Mining and Scientific Press*, August 10, 1912.
[7]*Ibid*, July 27, 1912. p. 118.

He shows how packed concentrate, settled into a cement-like mass, may be easily started up in a Pachuca by a method even simpler than that at the Alaska Treadwell referred to last year. In filter plants of the Butters type, where considerable storage room for pulp has to be provided, the provision of an efficient agitator in large mills has long been a pressing problem. The Australian type of agitator seldom exceeds 25 ft. diam. Hoyle's Esperanza agitator, a most likely type, could be 40 ft., or even 50 ft., but at the Modder B there is at the Butters plant a pulp stock-tank 60 ft. in diameter with smooth-running, under-driven, effective agitation, taking from 8 to 12 hp. at 4 r.p.m. This agitator (Central Mining & Investment type) is undoubtedly the best for the purpose which has yet come under my notice.

Amalgamation.—Since the Waihi G. M. Co. adopted amalgamation on plates set in a row away from the battery, this practice has progressed far and wide. In Africa the area of plates has been cut down by as much as 75%. Amalgamation now takes place only after tube-milling; the wear and tear of the coarsely crushed pulp on the after battery-plates is thus avoided. H. Stadler, who from time to time interjects some able criticisms on local practice, is not satisfied with the efficiency of the abbreviated plate area, but the Caldecott group is keen on the advantages they claim to thus derive. Undoubtedly the most daring practice is that introduced by Charles Butters at Nipissing.[8] No less than 40 tons of mercury is in circulation and the ore, containing native silver, argenite, and arsenites of cobalt and nickel, is ground in a tube-mill with mercury in greater weight than that of the ore charge and with 5% KCN solution. Air is introduced and a 98% extraction is reported in 9 hr. The silver in solution is precipitated by zinc shaving, and the mercury amalgam collected, retorted, and melted, with a resultant product of 999 fine. "It is well known that grinding in an iron mortar or pan with an excessive quantity of mercury will result in total extractions with even the most refractory ores,"[9] and it was this fact that induced the Cassel company, in the early days of cyanide, to standardize the two amalgamation tests ('ordinary'=plate amalgamation, and 'forced' =pan amalgamation) given on the page above quoted, as it was found that with other proportions the tests were liable to give results not comparable with those actually obtained in practice. In applying this procedure to general practice, I should have anticipated a large accumulaiton of sickened mercury. But sickened mercury may be retorted or even purified by other means, and it is to be hoped that ere long Mr. Butters will publish some description of the results of his undoubtedly bold, even if not ultimately successful, venture. Clark and Sharwood, in their valuable paper, already cited, refer to the excedingly slight penetration into the plate of the amalgam coating, and this is confirmed

[8]*The Mining Magazine*, June 1912, p. 452.
[9]James, Alfred, 'Cyanide Practice,' p. 33.

by McMiken, and Halse.[10] On the other hand, comment has been heard frequently on large amounts absorbed by plates, and as to plates being practically eaten through by amalgam. It would appear that unless plates are specially soft, the gold accumulation scarcely penetrates below the skin of the plate, even though the latter may be eaten through with mercury.

Classification.—W. A. Caldecott's cone and table, worked together, make probably one of the best products for cyaniding ever fed into a vat. In Africa, until recently, full advantage has not been taken of this fact, and even at the Simmer the product is fed with cyanide solution into collecting-vats and transferred into leaching-vats. It is obvious that with such product collecting-vats are an unnecessary expense (they already existed at the Simmer), and H. A. White, at the Princess, has shown how good are the results obtainable by direct treatment of the product. But cones are troublecome; they require continuous attention, and thus even the Caldecott table does not excite an enthusiasm for cones in comparison with the automatic simplicity and perfection of the continuous work of the Dorr classifier. But a further advantage of the Caldecott table is the dryness of the product—under 15% moisture—which enables the balance of solutions (15 to 20%) to be maintained with something in hand. Classifiers of the Dorr and Hoyle types just exceed the point, say 20 to 25%, and make draining the vats or some intermediate step necessary. Argall claims with his Ovoca classifier, working so successfully at Stratton's Independence, to give a drier product as the result of the compression of the pulp between his two spirals and to come just within the point, say 18 to 20%, and this type has accordingly been adopted in the most recent design of a large mill for Queensland.

Crushing.—The good results of the Nissen stamp have been confirmed in practice. New mines on the Rand, with their heavy stamps and open-front mortar-boxes, show, with the notable exception of the Roodeport United Main Reef mentioned above, an extraordinary vibration of cam-shafts, and in at least one case, of stems. The smooth running of the Nissen stamp at the City Deep has caused much favorable comment, and has undoubtedly been a factor in its adoption in the new Rhodesian mills. The tendency is to employ wider mesh screening. At the Modder B, one of the latest mills,, it is found that, with eighty 1600-lb. stamps, 8-mesh screening gives 11 tons per head per diem, and 3-mesh 16 tons. A useful table of variations in output for different mesh screens is given in 'Rand Metallurgical Practice,' Vol. I, p. 75. The most recent mill to start on the Rand is the Consolidated Langlaagte, with its 100 stamps of 1900 lb. each. It is probably the first mill in South Africa designed to produce more slime (60%) than sand (40%), and is thus a notable indication of

[10] *Trans.* Inst. Min & Met., Vol. XVII, p. 486.

the trend of local practice now that more efficient slime treatment (by filtration) is admitted to be available.

Tube-Milling.—Recent mills in Africa have the tube-mills set diagonally to the centre-line of the mill, to economize space and to afford a more convenient belt drive. The expensive Citroen gear of Deep City type, with its direct drive, has not been generally adopted. Belt drives appear to be preferred for the majority of the new mills. Ribbed liners of the Osborne type, and liners made up of concrete blocks studded with steel projections (old drill ends) are displacing silex. They are stated to have the advantage of longer life and of collecting amalgam scoured off the plates. Inded, the relining of a tube-mill is now found to add as much as 400 or 500 oz. to the output by the recovery of the amalgam collected by the old liner.

Authoritative tests will still be required on the loss in efficiency caused by the employment of ore in place of selected pebbles, and the money loss or gain resulting from such loss of efficiency as against the chushing in place of pebbles of some, it may be, 8000 or 10,000 tons of ore per year.

Filtration.—Announcements have been less assertive than usual. Merrill, Kelly, Sweetland, and Burt have not been so *much* discussed; the Moore and Butters filters about maintain their positions, with the Oliver coming into wider use in spite of the condemnation of its washing capacity expressed in Whitman Symmes' article, already cited. In Africa the good results of the Butters filter have undoubtedly caused excitement, even though they are not accepted as convincing by some engineers who are possibly at a loss to give valid reasons for their non-conversion. Expense of equipment is one of the excuses put forward; smallness of margin of recovery over decantation is another. An article in the *South African Mining Journal* of January 6, 1912, might therefore be referred to. At the Bantjes the extra cost was given as 3.4d. per ton treated, the extra extraction as 0.6 dwt., or say 2s. 6d., showing a gain of over 2s. per ton. This is, of course, an exceptional instance. At the New Modder, out of a total cost for slime treatment of 13¼d., the separate cost of the Butters is 4d. per ton, the total gain in extraction 11%, or say 8d., less 4d., equaling a gain of 4d. At the Crown Mines, for an extra cost of 3.6d. there was an extra extraction of 0.2 dwt., or 10d. From Australia come accounts of the Ridgway reciprocating unit. It apparently has been adopted by all the latest plants irrespective of groups, and by such well known, but differently controlled, companies as the Great Boulder, Associated Northern Blocks, and Sons of Gwalia. It claims a wide range of material from roasted sand to Bullfinch kaolin ore. It has been found there that the fixed submerged filters, when run on high-grade ore, tend to an extraordinarily great accumulation of gold in the wash water.

Heating Solutions.—W. A. Caldecott and W. R. Dowling heat solutions to 90 to 120° with waste steam at Knights Deep slime

plant, circulating them through two 6-in. centrifugal pumps at a cost of 25 hp. for 2000 tons per day. They claim good and clear settlement of solutions in the slime-plant, with a consequent increased (20%) tonnage treated.[11] A. H. Jones[12] shows the improved results obtained by him with solutions heated to 90 degrees.

Precipitation.—Reference is again made to Whitman Symmes' paper, cited above, for a note of his difficulties when using zinc dust and his method of overcoming them. The Merrill zinc dust method has not bulked so largely in the technical press of late. F. A. Vestal[13] gives his experiences at the Empire mines. Doubtless further progress will ere long be reported, although reversion to zinc shaving is reported by the Esperanza Mining Co., which after long use of the zinc dust method now passes its gold-bearing solutions, after they come from the Merrill zinc-press, through zinc shaving. It will be remembered that this is H. Colbath's practice at El Rayo, described in this review for 1910. A. J. Wyly refers to his precipitation at Simau with zinc shaving. One box containing 1200 lb. of zinc shaving precipitates daily 140 tons of solution containing 0.04 to 0.4% KCN, 0.6 to 20 dwt. gold, 1 to 10 dwt. Ag, so perfectly that the outflow assays 'traces' only.

Roasting.—It will have been noted that some of the latest Rhodesian plants are built on Kalgoorie roasting lines. The long hearth Edwards duplex furnace has been adopted for antimonial ores. This may be a grave mistake. The volatilized antimony tends to condense on the cold ore at the far end and thus to accumulate until the furnace is put out of action. It would appear better practice to choose a furnace in which provision is made for taking off the volatilized antimony before it comes in contact with the fresh charge.

General

During the year an attempt has been made in the *Johannesburg Star,* and later in the special anniversary number of the *South African Mining Journal,* to raise a fund for the benefit of those pioneers who, having spent the best part of their lives in introducing and taking charge of the working of the cyanide process on the Rand, now find themselves, after over twenty years' service, without that provision for their old age which they might have anticipated had the industry not succeeded in upsetting the MacArthur-Forrest patents and thus terminated the interest of the company which alone would have been able to grant pensions to the pioneers. It is not much to ask that in return for the free use of the cyanide process, and the special equipment, such as zinc boxes, designed for this purpose, a small sum per box used should be set aside as the nucleus of such a fund. It is stated that, not long since, an employee bearing an honored name in metallurgy, after some twenty years' service in perhaps the richest

[11]'Rand Metallurgical Practice,' Vol. I, p. 223.
[12]*Mining and Scientific Press,* Jan. 27, 1912. (Reprinted Page 87).
[13]*Ibid.* Nov. 9, 1912, p. 588.

and mine, was found dead the day after he had received notice to quit. It is hoped that this fund will prevent the recurrence of such an incident, which redounds neither to the honor nor the gratitude of the Witwatersrand gold industry.

MILL PRACTICE IN 1912

(Editorial, January 4, 1913)

Busy millmen find it increasingly difficult to follow the latest practice throughout the world, and the annual review of cyanidation during the past year by Mr. Alfred James, which appears on another page of this issue, is correspondingly welcome. To paint the rose is useless, but some of the interesting points touched upon invite further discussion. The successful treatment of rich silver ores calls for the most attention. At Tonopah splendid results are obtained at moderate cost from ores containing 30 ounces of silver and little gold. The mills at this centre are simple, and no complicated classification is necessary. The MacNamara and Belmont mills were started during the year, and the latter may be cited as an example of unusually good mechanical arrangement. It is making a high extraction at low cost, and in November produced 369,422 ounces, or about 12 tons, of silver bullion from 12,426 tons of ore. The Mexican mill at Virginia City was also started in 1912 and is making a 92 per cent extraction from an ore containing nearly 40 ounces of silver per ton. The application of hot cyanide solutions (about 60°F. at the mortar-boxes and 120°F. in the agitators) has been a great aid in securing higher extractions from these ores. At the Knight's Deep plant, on the Rand, solutions have been heated to 90 or 120°, but apparently this is to secure quicker settlement of slime, as no mention is made of increased extraction.

The 'new' metallurgy of Cobalt, in the mill of the Nipissing company, is of great interest. During November this plant treated 134 tons of a complex silver ore, yielding about 280,000 ounces of silver, or 2090 ounces per ton, with an extraction of 97 per cent. This is done in a tube-mill containing 3¼ tons of ore previously crushed to 20 mesh, 4¼ tons of mercury, 2 tons of 5 per cent cyanide solution, and 6 tons of pebbles. The pulp is filtered on vacuum-filters. This company has also built a stamp-mill for treatment of lower grade ores. Zinc shaving is generally preferred for precipitation of silver from cyanide solutions, but at the new Belmont mill zinc dust gives excellent results. Little has been heard from Mexico during the year, the revolution having interfered with the operation of many mills, but in his recent interesting little book, Mr. Ferdinand McCann gives details of the latest Mexican practice on silver ores.

It would seem that the so-called improved types of roasting furnace to which Mr. James refers are not well adapted to produce a sweet roast, such as is necessary in a dry treatment of sulpho-

telluride ores. Perhaps the most interesting roasting plant erected during the year is that at the Goldfield Consolidated mill, which will be described in an early issue of this paper by Mr. J. W. Hutchinson. This plant treats concentrate containing copper, and the method employed should be interesting to our friends at Kalgoorlie, where the roasting and treatment of dry ore and concentrate has reached a high point of efficiency, more especially at the Associated, Kalgurli, Perseverence, and the Horse-Shoe. Rhodesian ores contain antimony, and roasting is being adopted, but if the volatilized antimony oxide condenses and accumulates at the feed end of the furnaces, a modification of the construction will be required.

The gyratory crusher holds the field for coarse crushing in the majority of cases. Where a large capacity is required, this type is the best, but for small plants the jaw crusher is useful and suitable. The use of heavy stamps has its adherents, but greater attention is now paid to such primary crushers as Chilean, Lane, Krupp, and other mills and rolls. The new Buckhorn mill in Nevada will employ rolls, Hardinge mills, and tube-mills. Stage reduction is admitted to give best results, and rolls are now made so that grooving is largely prevented. In tube-milling a number of millmen believe in using ore or rock for crushing, though the efficiency of this is questioned by others, as rough surfaces do not facilitate good impact.

The methods employed in elevating pulp afford an interesting subject for discussion, but practice varies greatly. Bucket elevators work well in many mills, including the Goldfield Consolidated and all the Broken Hill silver-lead mills; centrifugal pumps are considered the best at Tonopah; taliing-wheels at Waihi and two Kalgoorlie plants; plunger pumps at Bendigo and Kalgoorlie; and pumps and wheels on the Rand. Agitation practice is equally varied. Vacuum-filters are being used in many new mills, but at Kalgoorlie the filter-press is doing its usual good work. Labor costs are, however, an objection to it, and probably the practice will have to go under in time. Results from the treatment of low-grade ores at Stratton's Independence, the Treadwell group, Homestake, Wasp No. 2, and the splendid work on ore worth less than $6 per ton at the Perseverance, South Kalgurli, and Associated Northern mills, in Western Australia, are all of the greatest interest. The use of oil-fired assay and clean-up furnaces is increasing, as they afford a particularly convenient means of applying heat. On the whole, the past year was one of great interest to all metallurgists, and further improvement may be confidently expected during the coming year.

INDEX

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